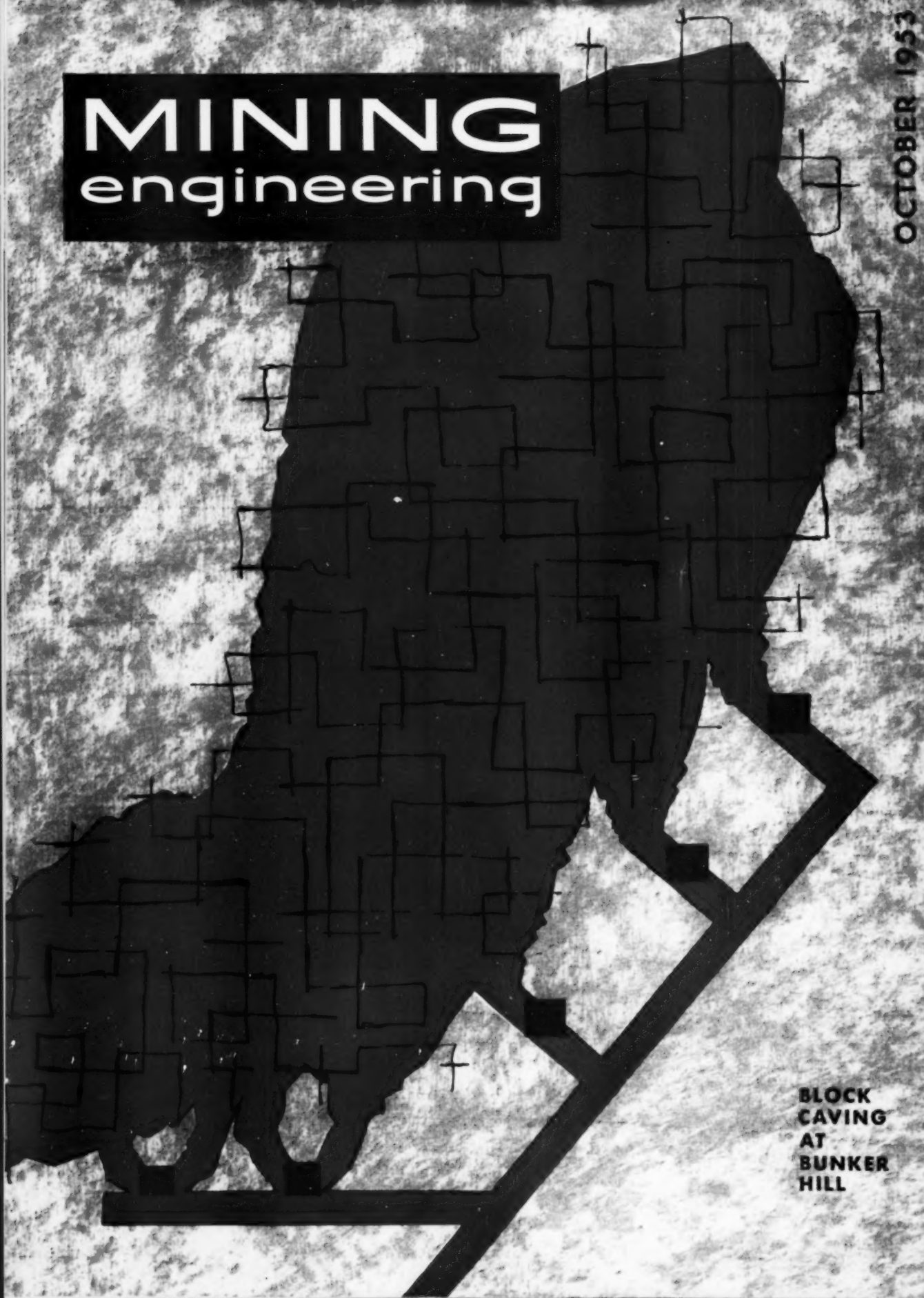
MINING engineering

OCTOBER 1953



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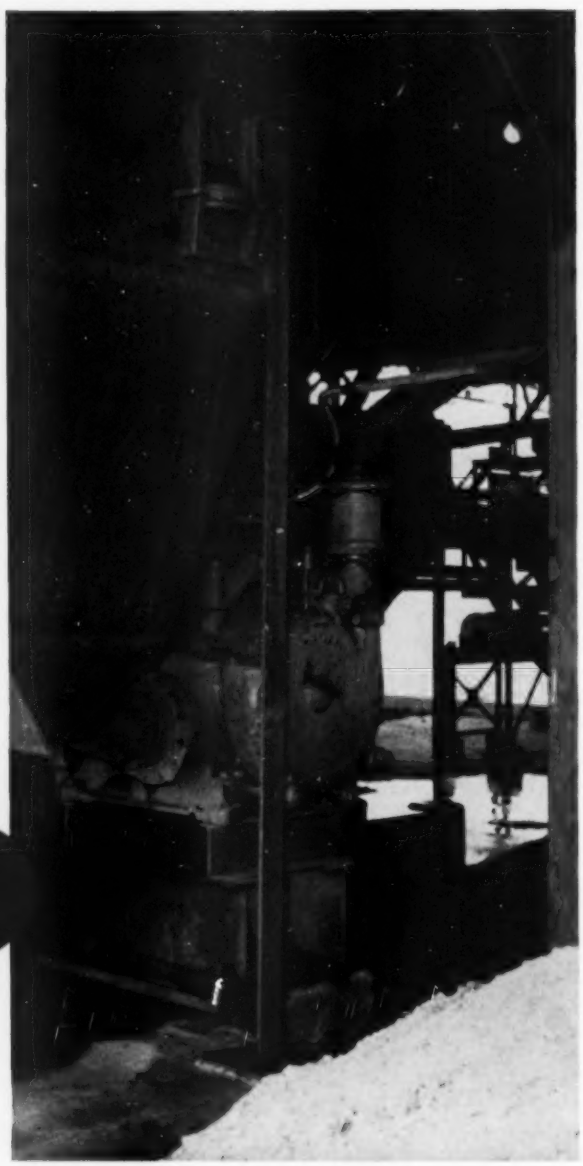

MINING engineering

VOL. 5 NO. 10 OCTOBER, 1953

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COVER

The red color on this month's cover dramatizes the shape of the orebody that Bunker Hill is mining by block caving. The slanted undercut, using shrinkage stoping, is not artistic license; the article starting on page 985 tells how and why it is done that way.

FEATURES

Personnel Service	942	Drift	973
Books	951	AIME News	1033
Authors	950	Personals	1042
Manufacturers News	956	Coming Events	1048
Reporter	959	Professional Services	1046
Mining News	963	Advertisers Index	1048
Trends	970		

ARTICLES

Induction Heating for Blast Hole Drill Bits — John H. Hearding, Jr. — 974

Summit Develops Pennsylvania Nonmetal Mine — 977

Open Pit Forum—
Carryall Scraper Plays Major Role in Open Pit Mining at Inspiration — Herbert H. Mellus — 978

Geochemical Prospecting Finds Widespread Application in British Columbia — Harry V. Warren and Robert E. Delavault — 980

What Automatic Controls Can the Mill Operator Use? — James E. Lawver — 982

TRANSACTIONS

Block Caving at Bunker Hill Mine — C. E. Schwab — 985

Alkali Reactivity of Natural Aggregates in Western U. S. — William Y. Holland and Roger H. Cook — 991

Orientation of Cube Diamonds in Drill Bits — Eugene P. Pfleider — 998

Fine Coal Drying — G. A. Vissac — 1004

Germanium and Other Elements in Coal and the Possibility of Their Recovery — A. W. J. Headlee — 1011

The Application of Forces to Centrifugal Gravitational Classifiers — Robert C. Emmett and D. A. Dahlstrom — 1015

Crushed Limestone Aggregates for Concrete — Katharine Mather — 1022

Recharging Ground Water Reservoirs with Wells and Basins — M. L. Brashears — 1029

—Personnel Service—

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

MEN AVAILABLE

Mechanical Engineer, 33, married, two children, AIME member. Eleven years responsible position metal mining, shops, diesel plants, design and layout underground and surface installations. At present employed mechanical superintendent large South American mining company. Seeking position in the States. German descent, not yet U. S. citizen. Available December 1. M-40.

Mining Engineer, 37, married. Fourteen years experience underground mining operations; exploration, development, mine valuation, and plant engineering. Registered Professional Engineer. Employed. Available reasonable notice. M-41.

Construction Project Manager, AIME member, college graduate. Would consider representing engineers or architects in the field during construction. Experience in hydroelectric projects, industrial and institutional buildings, beneficiation plants, and airports. Fluent Spanish. Foreign location desired, preferably Latin America. M-42.

Engineer, operation or consulting in copper smelter, 54, married. Twenty-five years operative and consulting in copper smelters and starting new smelters in Mexico, South America, South Africa. Available short notice. Location, immaterial. M-43-538-E-4 San Francisco.

POSITIONS OPEN

Mineralogist, preferably with Master's degree and one to five years experience in determinative mineralogy and related subjects for permanent laboratory position in research and development organization. Investigations in mineral separations and beneficiation problems also included. Good facilities available and real opportunity. Location, Pennsylvania. Y9018.

POSITION VACANT

Mining Engineer and Operating Assistant in Central Pennsylvania.

Write Box J-21 AIME
29 W. 39th St., N. Y. 18, N. Y.

Engineers. (a) Geophysical Engineer with about ten years broad experience in general geophysical exploration work, to work with staff doing mineral surveys in Iraq. Two year contract. Salary $13,000 a year plus living allowance and quarters, including transportation for employee and family to and from Iraq. (b) Mining Engineer, with about ten years experience in exploration, operation, and beneficiation of nonmetallics, preferably salt, sulphur, and potash, for work as above. Salary same as above. Two year contract. Location, Iraq. Y9017.

Mining Engineer, E.M., for general engineering having to do with quarrying, crushing and screening of rock and rock products for manufacturer of building materials. Must have had several years experience in quarrying hard rock, crushing and servicing and possibly some underground background. Knowledge of zeology. Salary up to $500 per month. Placement fee negotiated. Location, Chicago. C1242.

Mining Engineer, geologist, graduate, under 30, with 2 to 5 years experience for sales development assignment on jet flame piercing of blast holes. Laboratory tests, extensive field demonstration and installation. Experience in mineralogy, blasting practice, quarrying operations required. Salary open. Location, New Jersey. Y9084(a).

Research Associate, 25 to 30, with mining, metallurgy or geology degree, for ore dressing development project on laboratory scale. Salary, $4200 to $4800 a year. Location, New York, N. Y. Y9085.

Wanted by western non-metallic mine: Mine General Maintenance Foreman to direct and coordinate underground mechanical and electrical maintenance program. Must be familiar with the equipment of a fully mechanized mine. Applicants should give complete information as to age, experience and references. Salary open.
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3 Mining Engineers, young, with some full-time or summer experience preferable. Operation includes both open-pit and underground copper mines. Start at $4800 for six-day week. Kennecott Copper Corporation, Ray Mines Division, Ray, Ariz. Apply in person or by mail.

A recently built, modern operation in northern New York state is looking for a Graduate Engineer (preferably mechanical or chemical) with 5 to 10 years experience in the processing of nonmetallic ores or the manufacture of Portland cement. Experience in plant layout and design, familiarity with material handling. Salary open—reply giving references and resume of education and experience, and salary expected.

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1—MINING ENGINEER:

Qualifications: University Degree in Metalliferous Mining or similar qualification. Several years practical experience of exploration, development and operation in metalliferous mining, both underground and open pits. Some experience of heavy construction. Should have had administrative experience.

Duties: Examination and valuation of mineral prospects. Exploration of mineral deposits by drilling, boring, etc. Preparation of mine and plant designs. Supervision of construction work.

2—MINING ENGINEER (Coal):

Qualifications: University Degree in Mining or equivalent qualification. Several years experience of exploration, development and operation in coal mining. Adequate experience of coal washeries and carbonization, briquetting, calcining and coking plants. Should have had administrative experience.

Duties: Examination and valuation of coal prospects. Exploration of coal deposits by drilling, etc. Preparation of mine and plant designs for a coal mine under construction. Supervision of construction work on mine and plant.

3—NON-FERROUS METALLURGIST:

Qualifications: University Degree in Metallurgy or equivalent qualification. Adequate experience of methods of ore dressing, smelting and refining. Extensive plant design experience. Should be familiar with the use of electrolytic and electrothermic processes in the reduction of zinc and with oxidation and reduction of antimony.

Duties: Advice on methods of beneficiation of low-grade ores and on setting up of smelting and refining industries in zinc, tin, antimony and cobalt, and extraction of sulphuric acid from smelting and industrial gases. Setting up of pilot plants. Supervision of any construction work on smelting and refining plants.

4—ECONOMIC GEOLOGIST:

Qualifications: University Degree. Familiarity with modern methods of geophysical exploration. Adequate experience of calculation of ore reserves.

Duties: Reconnaissance surveys, exploration and geological mapping. Economic appraisal of mineral resources.

Apply to: THE DIRECTOR-GENERAL, MINERAL RESOURCES DEVELOPMENT CORPORATION, Secretariat, Rangoon, Burma, with references giving full particulars of age, education, qualifications and experience (in chronological order), stating salary and terms required.

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XHS3-5

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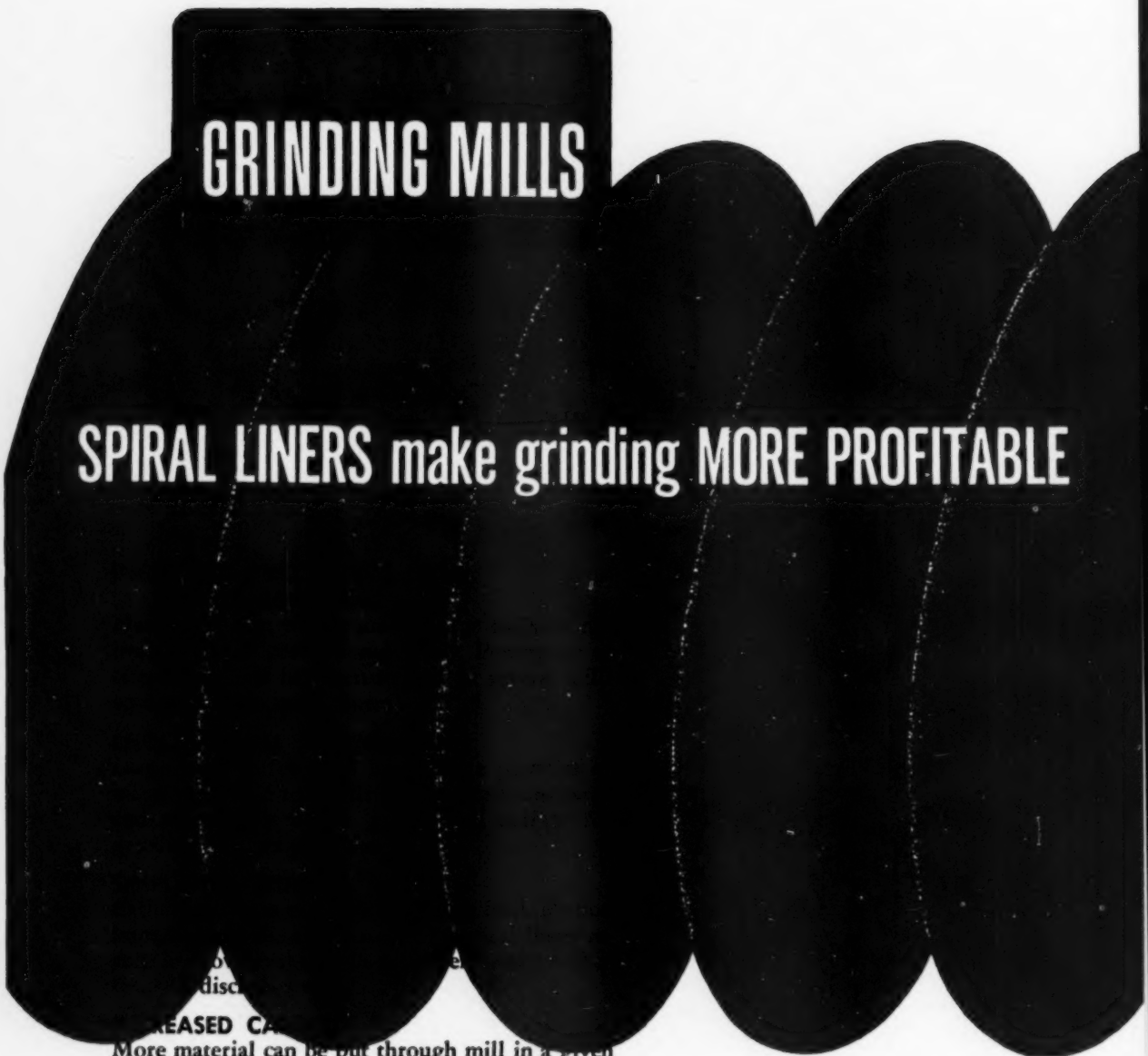

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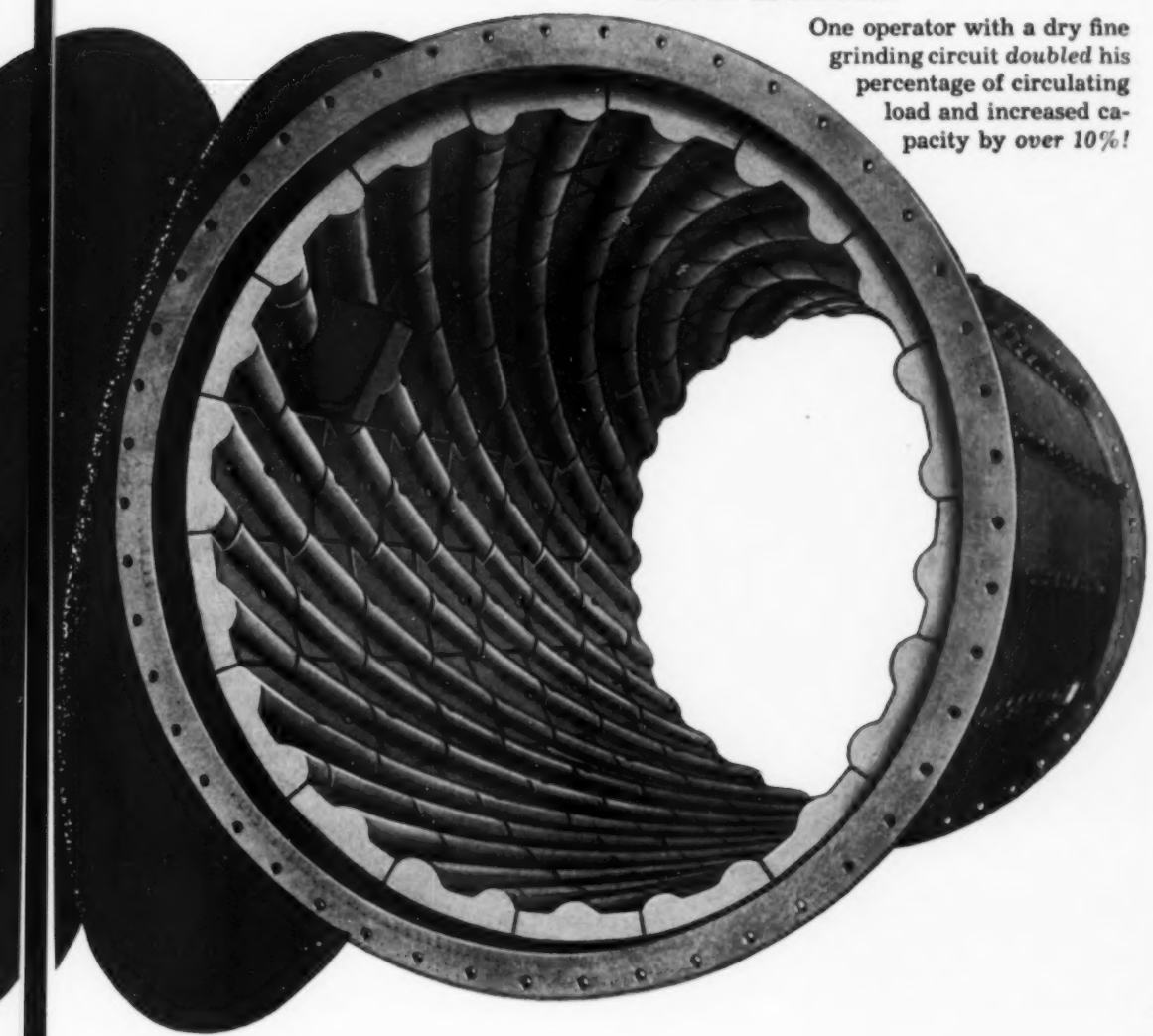
One operator with a dry fine grinding circuit *doubled* his percentage of circulating load and increased capacity by *over 10%!*



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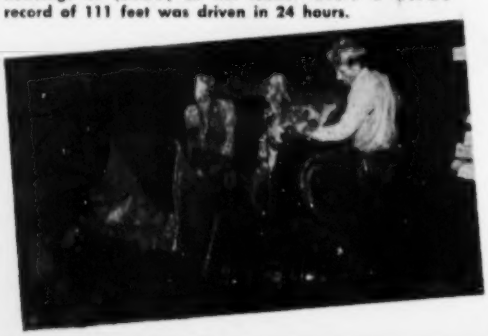

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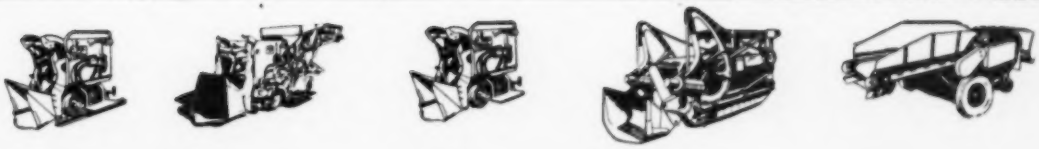

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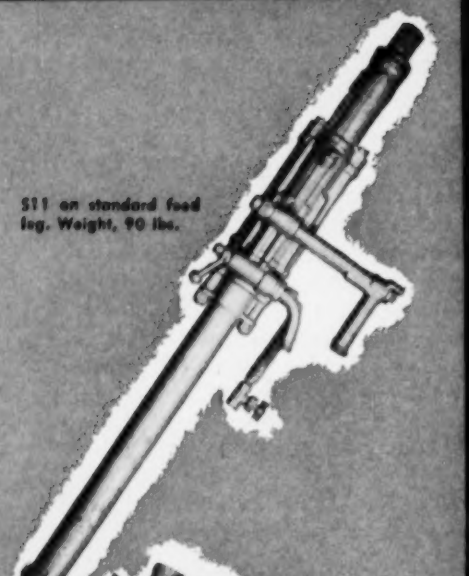
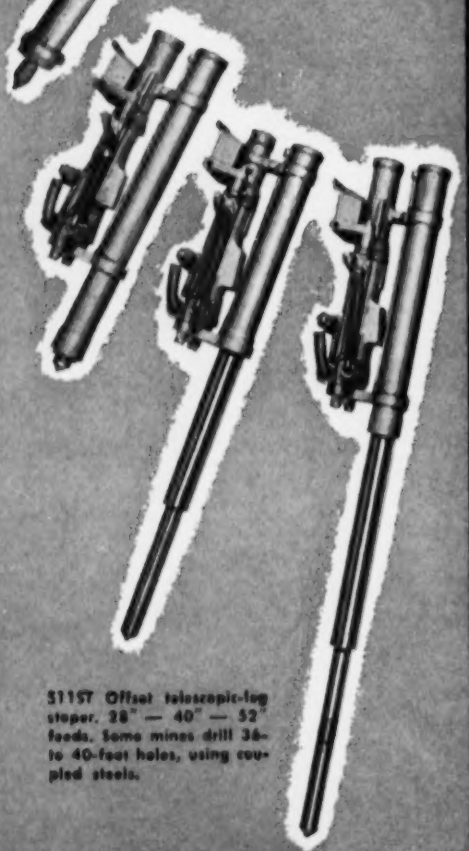


S11 on standard feed leg. Weight, 90 lbs.



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2. Air-feed relief control trigger is close to operator's thumb as he holds the handle grip. Slight pressure on the trigger eases up on the air feed — more pressure drops the drill back slightly, or returns it to its fully closed position.
3. Inside of air-feed cylinder is polished, to provide free action and insure longer life for cup leathers.
4. Trip rotation lets the operator stop and start rotation at will — a big help in collaring the hole. It's a safety factor, too: When a steel sticks suddenly, the pull of the rotation on the operator's hand turns the hand-grip sleeve, stopping the rotation before the machine can cause injury.
5. Air-feed piston is an integral forged tube — not welded or brazed.
6. Constant blowing around the steel keeps cuttings out of the chuck.
7. Rotating sleeve and spacer are built to withstand wear and fatigue.
8. Efficient lubrication protects all surfaces, adds to the dependability of Le Roi-CLEVELAND machines.

RD-24

MEET THE AUTHORS

James E. Lawver (P. 980) was born in Denver, Colo., May 28, 1919. After attending Colorado School of Mines from 1938 to 1940 he went to work for Basic Ores Inc., Lunning, Nev., as a junior engineer. He returned to Colorado School of Mines in 1941 and received a degree in metallurgical engineering in 1943. Following graduation he was a level foreman for Braden Copper Co., Sewell, Chile. In 1945 he was with the Foreign Economic Administration in Bahia, Brazil. That same year Mr. Lawver returned to the U.S. and joined International Minerals and Chemical Corp., at Mulberry, Fla., as a research engineer. He is currently a graduate student of metallurgy at the Colorado School of Mines. Mr. Lawver became an AIME member in 1946 and was Chairman of the MBD Operating Control Committee in 1952.

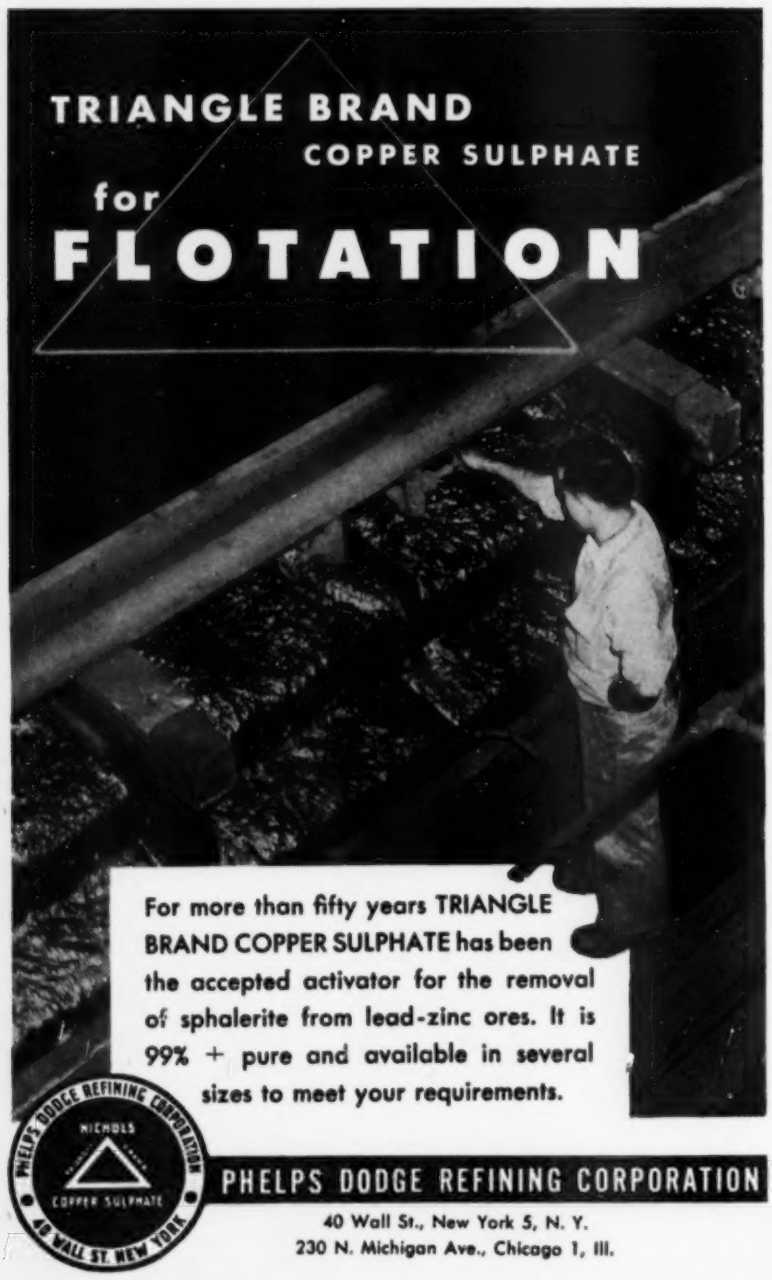
M. L. BRASHEARS

M. L. Brashears (P. 1029), partner in Leggett & Brashears, consulting ground water geologists, is a graduate of the University of Maryland and MIT, earning a Bachelor's in geology in 1933. As an undergraduate he played football, ran on the track team, and belonged to Sigma Nu. Mr. Brashears was employed by Lawton Engineering Corp. as an air conditioning engineer. Later he joined the U. S. Geological Survey as district geologist for New York and New England ground water operations. Mr. Brashears has authored several papers. Stamp collecting, woodworking, and a golf game in the 90's take up his spare time.

G. A. VISSAC

G. A. Vissac (P. 1004), has been the author of several papers on coal cleaning and drying for the AIME. He has been a member of the Institute since 1927. Born in France, Mr. Vissac holds a Bachelor of Latin Sciences, Master's Degree in Mathematics, and is a graduate mining engineer. He attended the Ecole Nationale Superieure des Mines in south east France. Mr. Vissac was employed as mine manager and assistant to manager, Carmaux Collieries, France. In 1921 he went to West Canadian Collieries Ltd. in Blairmore, Alberta. He is now a consulting engineer and member of the Dominion Coal Board.

Correction: The aluminum mine car shown on page 759 of the August MINING ENGINEERING is described as being built by Reynolds Metal Co. The car was designed by Enterprise Wheel and Car Corp. in conjunction with Reynolds Metal Co., and built by Enterprise.—Ed.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Books for Engineers

Recent Developments in Mineral Dressing, Proceedings of a Symposium on Mineral Dressing held at the Imperial College of Science and Technology, London, September 23 to 25, 1952, *The Institution of Mining and Metallurgy*, London, $8.50, 700 pp., illustrated, 1953.—This volume contains the Third Sir Julius Wernher Memorial Lecture by Prof. A. M. Gaudin, *Radioactivity in Mineral Dressing*, together with 39 papers presented at the Symposium, a report of discussion, contributed remarks, and index.

Clay Mineralogy, by Ralph E. Grim, *McGraw-Hill Book Co. Inc.*, $9.00, 334 pp., 1953.—This new text summarizes available data on the structure, composition, properties, occurrence, and mode of origin of various clay minerals.

Geological Investigations of Strontium Deposits in Southern California, by Cordell Durrell, *California Department of Natural Resources, Division of Mines*, Special Report 32, $1.25, 48 pp., 9 pls., 12 figs., 1953.

Sulphur. System No. 9, Section A2 and B1, of the eighth edition of Gmelin's Handbook of Inorganic Chemistry, Edited by the Gmelin Institute. *Verlag Chemie*. A2 450 pp., BI 372 pp., 1953.—The volume A2 treats, for the first time in modern scientific literature, the complete knowledge of the occurrence and technology of the sulphur compounds, of the colloidal chemistry of sulphur and the physiological phenomena caused by sulphur. Section B1 deals with sulphur hydrites and oxides. Formation of hydrogen sulphide from the elements, the equilibrium, and the composition of H_2S are fully discussed. Part A of System No. 9 will be concluded by Section A3 to appear in the fall 1953 and Part B will be completed by Section B2 in 1954. (A free booklet describing the Gmelin Handbook and listing the books available in this famous German series is available from Walter J. Johnson Inc., 125 East 23rd St., New York 10, N.Y.)

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Recent U. S. Patents

2,633,987 Viscosity control in heavy-media separation. (4/7/53) *American Cyanamid Co.*

2,634,198 Coal carbonization and gasification process. (4/7/53) *Hydrocarbon Research Inc.*

2,634,204 Pyrometallurgical process for manganese recovery. (4/7/53) *Sylvester & Co.*

2,634,915 Centrifugal impact pulverizing apparatus. (4/14/53) *Paramount Mining & Milling Co.*

2,635,036 Treatment of titaniferous ore to produce $TiCl_4$. (4/14/53) *The Dow Chemical Co.*

2,635,957 Method of gas control in ore reduction. (4/21/53) *Stora Kopparbergs Bergslags*, Sweden.

2,636,604 Flotation of pyrite (4/28/53) *Bethlehem Steel Corp.*

2,636,806 Acidulation of phosphate rock. (4/28/53) *Tennessee Corp.*

2,636,924 Method of geophysical exploration. (4/28/53) *Lundberg Explorations Ltd.*

2,637,545 Manufacture of magnesia and calcium carbonate from dolomite. (5/5/53) *M. Hicguet*, France.

2,638,214 Jig for ore and coal treatment. (5/12/53) *E. Harvengt*, Belgium.

2,638,402 Seismic prospecting apparatus. (5/12/53) *The Texas Co.*

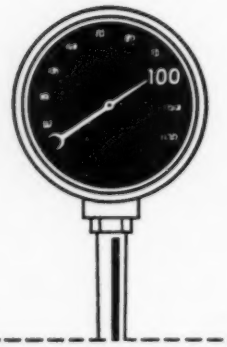
2,639,220 Method of making copper sulphate from copper ores. (5/19/53) *A. M. Thompson.*

2,639,974 Uranium recovery by smelting. (5/26/53) *A. Rose, R. Farmer, A. Lamont*, Canada.

2,639,975 Preparation of white zinc oxide from carbonate zinc ores. (5/26/53) *F. Coppet, R. Saffores*, Switzerland.

Purchase U. S. Patents ($0.25 ea.) from: Commissioner of Patents Washington 25, D.C.

keeping air pressure UP THOUSANDS of feet DOWN



At the Silver Summit Mine in Osburn, Idaho, in a vaulted room more than 1,000 feet below the surface, an Ingersoll-Rand PRE compressor (shown below) converts electric power into air power used to help tear silver, copper and lead ores from the earth.

This spic and span installation, shown below, is reached through a tunnel 6,000 feet long. The 400-hp compressor provides air for mine workings as far as 3,400 feet below it.

Placing the compressor underground has two advantages: First, it eliminates costly piping that otherwise would be necessary to transmit compressed air more than a mile from a surface plant to the working shaft. Second, it avoids air pressure losses that would be caused by leakage and the friction of air passing through a mile-long pipe.

This is not the first underground PRE installation for the Hecla Mining Company whose management controls the operation of Silver Summit Mining Co. It had previously relocated one unit at its Star Mine from the surface plant to an underground station where it also installed a new PRE. This step eliminated the surface plant and the pressure losses formerly caused by 11,000 feet of air piping. Several other I-R compressors have been serving Hecla's older properties for more than 30 years.

Long recognized as a leader in using modern mining equipment to increase production, Hecla mines employ many Jackhamers, drifters, Jackbits, hoists, pumps, and other I-R products.

861-1

FEATURES OF THE "PRE" COMPRESSOR

- Crankcase need not be opened for bearing adjustments . . . stays clean.
- Main bearings have micrometer adjustment from outside of crankcase.
- Full-floating crankpin and crosshead-pin bearings result in long bearing life.
- Unique, built-in, force-feed running gear lubrication eliminates piping both inside and outside compressor.
- I-R stainless-steel Channel Valves are efficient and durable.
- Automatic 5-step full Clearance Control regulates compressor output.
- Built for continuous full-load operation . . . for air or any gas . . . for any pressure . . . sizes from 300 to 3000 hp.

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GM DIESEL
CASE HISTORY NO. 1A3-12

OWNER: Georgia Coating Clay Co., Macon, Ga.

INSTALLATION: Twelve GM Diesels power shovels, Koehring Dumptors, GMC Trucks stripping 97 feet of overburden and mining Kaolin.

PERFORMANCE: Equip. Supt. W. J. Herrington reports 6-71 powering Koehring 1½-yd. shovel (background) has worked 45 hours a week, 52 weeks a year for over 6 years--about 14,000 hours--without an overhaul. Newer ¾-yd. Lima (foreground), powered by a 3-71, has already operated for 3500 trouble-free hours.

GM
GENERAL MOTORS
DIESEL POWER

14,000 HOURS
WITHOUT ENGINE OVERHAUL

Long engine life between overhauls is only one advantage you get with General Motors Diesel engines. Power on every piston downstroke makes a 2-cycle GM Diesel accelerate faster under load—gets more work done every hour. Simple, practical design cuts down time because it does away with such trouble sources as high pressure fuel lines. When service is needed, you'll find the cost surprisingly low. GM Diesel engines are easy to work on—simple to inspect and service. And comparison will show you that GM Diesel parts cost less.

You can get GM Diesel power in leading makes of shovels, tractors, hauling units, loaders, compressors and generators. A postcard will bring you a list of the 750 different models of equipment powered by General Motors Diesel engines.

DETROIT DIESEL ENGINE DIVISION
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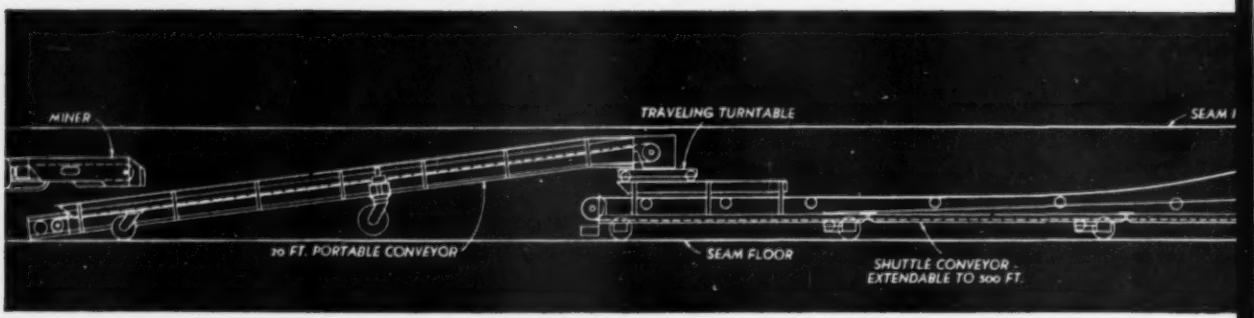
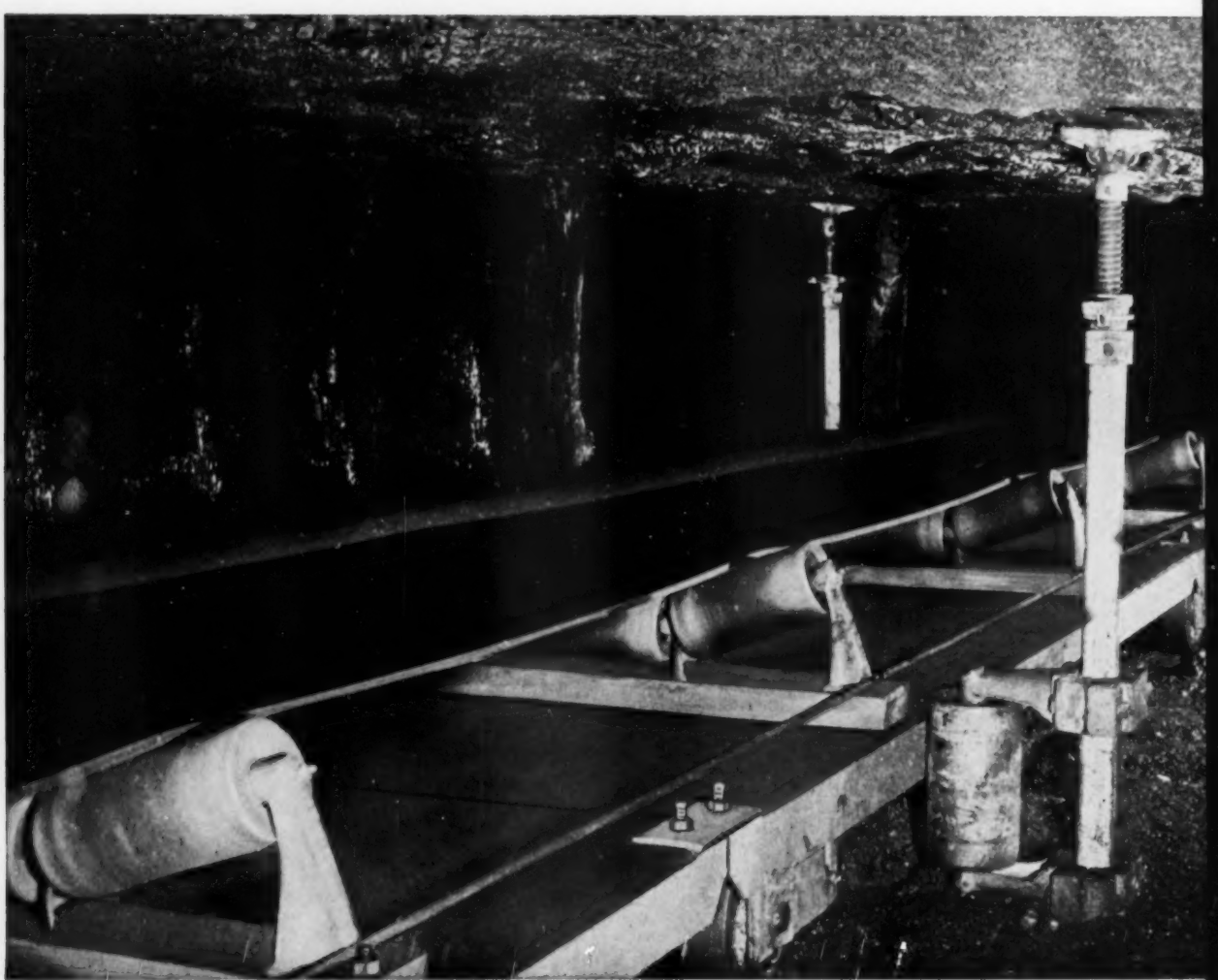
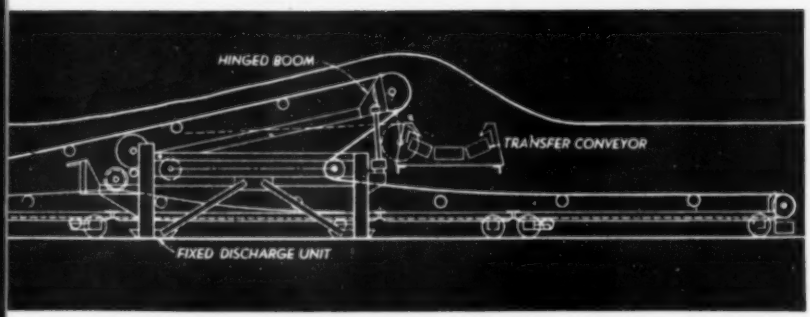


Here's the New Answer to

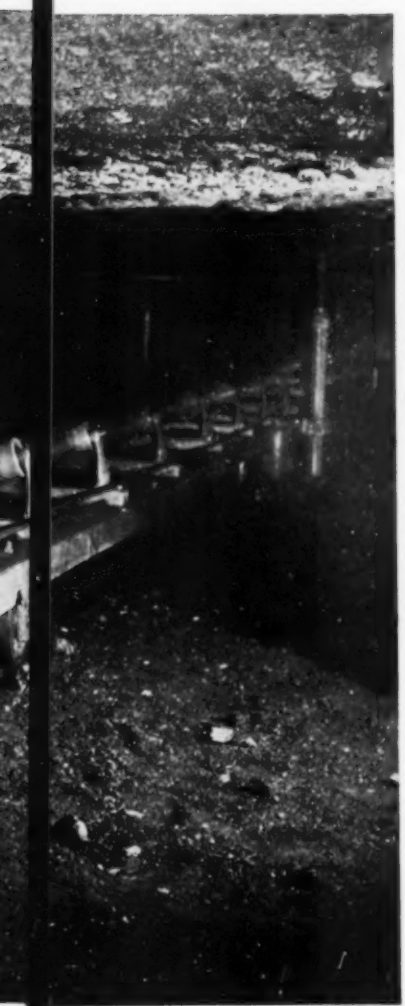


HEWITT-ROBINS

EXECUTIVE OFFICES, STAMFORD, CONNECTICUT



Continuous Mining



It's The Hewitt-Robins Mine Type Shuttle Conveyor With Fixed Tripper

The Hewitt-Robins Mine Type Shuttle Conveyor with fixed tripper forms the final link in a complete belt conveyor system that can handle an uninterrupted flow of ore directly from the face to the surface as fast as any mechanism can produce it.

This versatile unit is the key to truly continuous mining. The Shuttle Conveyor is both extendable and retractable —can follow the progress of mining and at the same time maintain a fixed transfer point through a fixed tripper and is extendable to 600 or 700 feet.

As the working face advances, the Shuttle Conveyor can closely follow the mining machine and receive its load either directly or from an intermediate transportation unit such as shuttle car or loading machine. The Shuttle Conveyor is so designed that alignment can be properly maintained by mounting small guide rollers on standard roof-jacks along each side of the conveyor frame, when operating off-track. When track-mounted, the guides are not required.

Remember, when it comes to any type of belt conveyor or complete belt conveyor systems, only one company—Hewitt-Robins—can assume single and unified responsibility for successful operation. For only Hewitt-Robins designs, engineers, manufactures and installs both the belt and machinery.

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☐ —Slope ☐ —Vibrating
☐ —Fixed Tripper Shuttle

BELTING:

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*☐ —Hot Materials
*☐ —Raynile®
*☐ —Steel Wrapper
*☐ —Transmission
☐ —Woven Wire
☐ BUCKET ELEVATORS
☐ IDLERS

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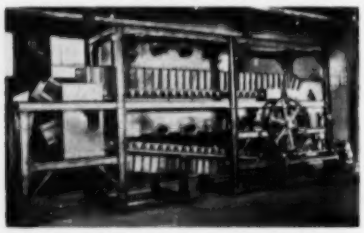
Manufacturers News

New Products • FILL OUT THE POSTCARD FOR MORE INFORMATION • Equipment

Pilot Sintering Machines

A new line of Dwight-Lloyd pilot sintering machines built by *Sintering Machinery Corp.* offers features of larger machines and is said to provide flexible units on which process requirements can be worked out



with certainty that commercial machine results can be directly interpreted. The 24 in. x 9-ft machine is illustrated, and other models range from the 12 in. x 4-ft research machine to the 36 in. x 30-ft research-pilot unit. **Circle No. 1**

Picturesort

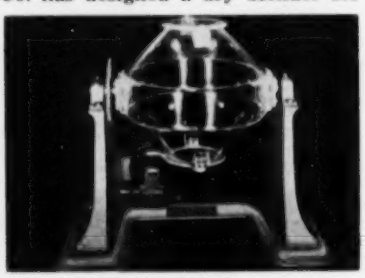
A means of attaching microfilm strips of actual patents to file or punch type patent abstract cards has been developed by *Picturesort Co.* Used with the 1.4 million patent abstract cards company has available, the system can be adapted to other records. **Circle No. 2**

Concrete Drill

A water feed attachment which fits garden hose connections for use with *Tilden Tool Mfg. Co.'s* Rotary Konkrete Kore drill is asserted to prolong drill life and increase drilling speeds up to 300 pct. **Circle No. 3**

Laboratory Blender

Patterson Foundry and Machine Co. has designed a dry blender for



research and development use. Clear plastic shell has 0.31-cu ft capacity. **Circle No. 4**

Reusable Thimbles

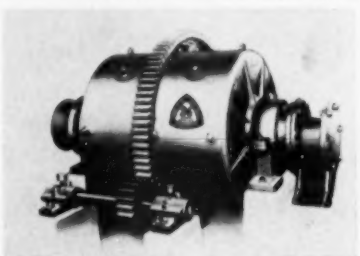
An 8-part braided sling introduced by *A. Leschen & Sons Rope Co.* features a reusable thimble fitting reported to save sling users up to 20 pct or more. The Pin-Lock thimble, attached by pins instead of conventional hammered-down clamps, is readily reuseable. **Circle No. 5**

Dust Measurement

Mine Safety Appliances Co. has been named U. S. distributor for dust measuring instruments manufactured by *C. F. Casella & Co.*, London. MSA-Casella instruments now being introduced to American users include: Cascade impactor, for discriminatory collection of dusts by particle size; the Settlement dust collector; the Thermal Precipitator for sampling smoke, fume, and dusts; and the Jet dust counter for rapid sampling without interruption of operations. **Circle No. 6.**

Economy Ball Mill

Denver Equipment Co. is producing a 30-in. economy ball mill. The low-cost 30x18-in. size sells for $785



f.o.b. Denver, and the 30x36-in. mill sells for $1195. The 30x18 mill can be enlarged to 30x36. **Circle No. 7**

V-belt Fastening

Package introduced by *Flexible Steel Lacing Co.* makes up V-belts of any length for use in emergencies and when correct endless belt is not available. **Circle No. 8**

Automatic Filling Scale

Completely automatic operation is feature of *Thayer Scale & Engineering Corp.* unit that automatically holds, fills, and check-weighs bags of powdered materials. **Circle No. 9**

Flow Detection

Detect-A-Flo, precision air flow instrument, has been introduced by *Fenwal Inc.* for applications in ventilation control, protection of air-cooled equipment, and use as a liquid-level detector. **Circle No. 10**

Butterfly Valve

Keystone Tool Corp. recommends the Keystone Butterfly Valve for use on liquid, sludge or dry material lines. Valves are made in sizes from 2 to 12 in. Handle mechanism is indexed to hold the valve. **Circle No. 11**

H. J. Daniels & Co.

According to *H. J. Daniels & Co.*, Hazelton, Pa., future production of DMS (Dense Media) coal preparation equipment will be in its own shops. Sales offices will be available in principal cities. **Circle No. 12.**

Blacklight

Menlo Research Laboratory has developed the compact Blacklight Wand portable ultraviolet instru-

ment. Total weight with flashlight batteries is only 1¾ lb, but construction is said to permit rugged field use. **Circle No. 13.**

Fire Extinguishers

New action labels picture what to do and how to do it on *American-LaFrance-Foamite Corp.'s* fire extinguishers. Even one who cannot read can properly operate extinguishers by studying action sketches. **Circle No. 14.**

Coal Drying

An automatic control system for uniform drying of washed coal, designed by *The Foxboro Co.* for the McNally-Vissac coal dryer, regulates temperature so that variations in feed rate and initial water content do not affect final shipping weight. **Circle No. 15**

Scrapers

A new line of scrapers ranging in capacity from 10.5 to 28.5 cu yd was announced by *LeTourneau-Westinghouse Corp.* Units incor-

porate welded construction throughout and the double drum power control unit cables work through self-aligning sheaves. **Circle No. 16**

Crushing Plants

Portable crushing and screening plant announced by *Diamond Iron Works Inc.*, is rated at over 350 tph

of —¾ in. aggregate, and features 36x22 and 10x36-in. crushers, 4x12-ft vibrating screen, and 30-in. conveyor belts. **Circle No. 17**

Machinery Center Inc.

The Roger Pierce Equipment Sales Co. of Salt Lake City has announced change of its corporate name to *Machinery Center Inc.* The officers and mailing address remain the same.

Free Literature

(21) VIBRATING SCREEN: New booklet describes *Hewitt-Robins* style J Vibrex vibrating screens, recommended for use in small quarries and contractor's plants.

(22) TAPER-LOCK: Available from *Morse Chain Co.* is a 16-page, pocket-size catalog listing prices and specifications for Morse Taper-Lock stock roller chain sprockets.

(23) PIPE REPAIR: *Morris Coupling and Clamp Co.* tells how pipe line leaks can be repaired quickly with Morris Band Type Repair Clamps.

(24) FUME KILLER: OCM Catalytic Exhaust, a device for removing carbon monoxide, fumes, and odors from all types of gasoline-powered equipment, is illustrated and described in folder by *Oxy-Catalyst Inc.*

(25) WRENCH: *Blackhawk Mfg. Co.* says its Wedg-Head wrench fits into tight corners, against walls, and over obstructions where sockets, box wrenches, and other ordinary open ends can't reach.

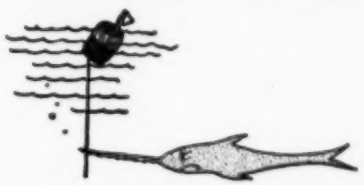
(26) GAS ANALYSIS: The new 48-page *Burrell* catalog lists and describes equipment and accessories for gas analysts.

(27) SPEED REDUCERS: A 15-page booklet on *Westinghouse Electric Corp.'s* complete line of speed reducers discusses equipment features, manufacturing techniques, special units developed for right-angle drives, in-line drives, etc.

(28) GENERATING PLANTS: A pocket-size booklet from *D. W. Onan & Sons Inc.* traces history of electric plant development; describes ac, dc, and battery charging plants; discusses cost of operation.

(29) ELECTRONIC RECORDER: Simplified design, accuracy to ±1 pct of full scale, self-starting operation from low energy sources are the features of Tektolog electronic recorder described in an 8-page technical data bulletin from *Fielden Instrument Div.*, Robertshaw-Fulton Controls Co.

(30) WIRE ROPE: A steel core of 250,000 psi tensile strength, a hard jacket of transparent nylon, protecting the core from corrosion and increasing fatigue resistance up to five times, are some of the properties and qualities of Wirlon made by *Rochester Ropes Inc.*



(31) PREVENTIVE MAINTENANCE: *Eutectic Welding Alloys Corp.* claims that Eutec-Instant-Overlay can be painted on metal surfaces and after application of torch or carbon arc to melt the paste, bonds to the metal to form a "skin thick" film of hard, wear resistant, overlay (up to 65 Rockwell C).

(32) FLAMEPROOFING: An oil base fire retardant interior flat wall paint is claimed to contain a mass of minute "built in" fire extinguishers. When exposed to flame Fyr-Kote, manufactured by *Morris Paint & Varnish Co.*, pours out carbon dioxide and calcium chloride which smothers fire and retards spread of flame. Listed by Underwriters' Laboratories.

(33) TAPE REEL: *The Lufkin Rule Co.* has a new five-armed reel for holding long narrow steel measuring tapes. Descriptive folder illustrates, and gives prices for complete line of Lucas mine tapes.

(34) PNEUMATIC SAW: *Wright Power Saw and Tool Corp.* announces the New Model AS-221 pneumatic Wright saw. Employing the exclusive reciprocating double sawblade principle, this new model is said to cut in the hardest-to-get-at places, saw a smooth finish along a chalkline, crosscut, rip or angle through an 18-in. timber, and saw circles with a 12-in. radius.

(35) CHAIN HOIST: The Pul-Lift, an unusual chain hoist design which either hoists or pulls, is illustrated and described in bulletins from *Yale & Towne Mfg. Co.*

(36) MOTOR STARTERS: Features of Type H starters for motors in ratings from 2200 to 5000 v are described in bulletin from *Allis-Chalmers Mfg. Co.*

(37) DIAMOND DRILL BITS: Excellently illustrated, a 16-page booklet issued by *Koebel Diamond Tool Co.* aids in selecting proper bit for each job, describes powdered metal matrix, and shows available diamond carrying sections for a patented insert-type core bit.

(38) WELDING MACHINES: Four ac and two dc welders built by *A. O. Smith Corp.* are illustrated in brochure giving features of newly designed line.

(39) SLIDE PRESENTATION: "How to make your own slide film presentation for less than $20," available from *Minnesota Mining & Mfg. Co.*, tells how tape recordings and

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for more information on items described in Manufacturers News and for bulletins and catalogs listed in the Free Literature section.

Mining Engineering **29 West 39th St.** **New York 18, N. Y.**

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color slides can be combined for profitable application in employee indoctrination and training, for sales meetings, and for presentations.

(40) HAND WIPER: Dermatitis in 1952 in industrial plants was responsible for an estimated loss of 2.25 million man days of work. This,

Scott Paper Co. believes, is likely to be greatly reduced with a new industrial wiper made of two "Perf-Embossed" sheets, welded together for extra durability.

(41) WELDING TECHNIQUE: *International Nickel Co.* has issued a detailed 44-page study of various welding techniques for nickel alloy steels, including data on newer inert gas processes, electrodes, and heat treatments.

(42) POWER FOR COAL MINING: "Economical Power" is title of illustrated pamphlet from *Caterpillar Tractor Co.* showing engines at work in strip and underground coal mines.

(43) CONTROL INSTRUMENTS: Complete engineering specifications and prices of controls for furnaces, ovens, dryers, and kilns have been provided in a *Bristol Co.* catalog.

(44) PLANT EQUIPMENT: Plants and equipment for the phosphate and ammonia fertilizer industry are covered in a book released by the *Link-Belt Co.* Recent improvements in processing technique are illustrated in perspective flow diagrams.

(45) MATERIALS HANDLING: New methods in bulk and packaged materials handling to reduce operating costs for industry are pictorially presented in "Modern Materials Handling" brochure from the tractor division of *Allis-Chalmers Mfg. Co.*

(46) DRYERS: *Denver Equipment Co.'s* Standard dryers are described in a new bulletin listing features of installation, drive, flight, rolls, and seals, and giving typical drying data for various materials in dryers 2x12 to 10x100 ft long.

(47) PAINTING: Information sheet from *Carboline Co.* promotes no particular brand of paint, but gives data for buyer and maintenance engineer on basic economy in purchasing and using paint for corrosion protection.

(48) RAILROAD DUMP CARS: Bulletin released by *Baldwin-Lima-Hamilton Corp.* illustrates mechanical design of Western air-operated dump cars built in 30 and 50 cu yd capacities.

(49) PIPING AND PRESSURE VESSELS: *Taylor Forge and Pipe Works Inc.* has new publication available to people interested in piping and pressure vessels.

(50) COST CUTTING: Revised edition of "The HOW Book of Cost-Cutting Materials Handling" is available from *Yale & Towne Mfg. Co.* The 28-page book gives methods for simplified fact collection on materials handling operations and illustrates efficient application of handling equipment.

(51) RUBBER BEARINGS: Uses and advantages of Cutless rubber bearings for pumps, agitators, and other equipment where bearings are lubricated by water or the liquid handled, are outlined in catalog from *B. F. Goodrich Co.*

(52) CONTROLLED PUMPING: Full story of *Milton Roy Co.'s* controlled volume pumps for precise metering of liquids in measured volume with accuracy within 1 pct, is told in 36-page booklet that gives operating features, specifications, and materials selection charts for special pumping problems.

(53) MICROPHOTOMETER: Complete information on Knorr-Albers recording type and Vincent-Sawyer spotting microphotometers is in revised catalog issued by *Leeds & Northrup Co.*

(54) STEEL PIPE & TUBE: Condensed technical data on Croloy pipe and tubing for high temperature service is provided in folder from *Babcock & Wilcox Co.*

(55) STORAGE SHELTER: Unique section storage shelter made in independent sections to be mounted on a track to form a building is described in literature from *Yard-Stor Shelter Co.* New shelter building offers complete flexibility, easy movement in sections by crane or truck. Individual sections find use for construction projects, tunnel openings, etc., to provide all-weather protection for men, equipment, and materials.

(56) CENTRIFUGAL BLOWERS: A 32-page bulletin from the *Lamson Corp.* catalogs centrifugal blowers and exhausters for air and gas handling. Technical data on use of air for agitation of liquids, combustion, and pneumatic conveying are provided, together with fundamental engineering laws for blower operation.

(57) HYDRO-CLASSIFIER: Bulletin from *Hardinge Co.* tells story of the

hydro-classifier as a large volume unit for solving fine size separation problems.

(58) COST CUTTING CONTROLS: *Photoswitch Inc.* announces a new edition of "Cutting Production Costs with Electronic Controls"—said to be a "Bible" of electronic control for engineers, production executives, and management personnel. The 46 case studies explain how packaged units solved control problems.

(59) WELDING PROGRESS: Eighteen new welding developments have been announced by *Eutectic Welding Alloys Corp.* in the format of a digest type magazine.

(60) INDUSTRIAL CONTROL: An 80-page catalog from *Minneapolis-Honeywell Regulator Co.* covers non-indicating electric, electronic, and pneumatic controllers for temperature, pressure, and humidity.

(61) WIRE ROPE RECOMMENDATIONS: "Wire Rope Recommendations for Mining Industries" is well-illustrated booklet issued by Hazard Wire Rope Div. of *American Chain & Cable Co.*

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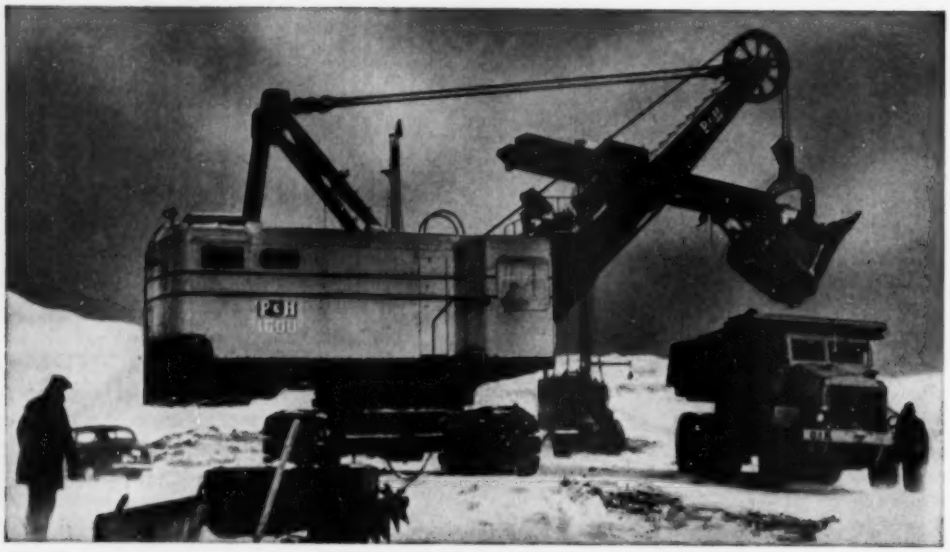


Power Shovel... equipped with dipper sticks and other heavily stressed parts fabricated from U.S.S. Tri-Ten, a manganese-nickel-copper steel produced by UNITED STATES STEEL COMPANY.

Characterized by exceptional toughness, Tri-Ten steel withstands repeated shocks, even at sub-zero temperatures. Its fatigue resistance is 50% higher than that of structural carbon steel. Yield point is 50,000 p.s.i., and tensile strength is 70,000 p.s.i., minimum, in ½" thicknesses or under. These values are moderately lower for thicknesses up to 4".

Very good weldability in intermediate and heavier sections makes Tri-Ten steel especially suited for fabricating heavy-duty equipment requiring maximum ruggedness and stamina with minimum weight . . . as demonstrated by this shovel, manufactured by HARNISCHFEGER CORPORATION, Milwaukee, Wisc.

How Nickel helps equipment stay on the job

...At 40° Below Zero

Remember when insistent demands for iron ore made it necessary to work Minnesota's Mesabi range in winter?

Power shovel operators ran into plenty of trouble at that time...for when they rammed their huge steel dippers into ore frozen solid by sub-zero temperatures, booms and dipper sticks broke all over the range...

With one significant exception, however...

Shovels equipped with dipper sticks fabricated from U.S.S. Tri-Ten... a steel toughened by the addition of nickel and other alloying elements... went right on loading the frozen ore. In fact, a leading shovel manufacturer reports that Tri-Ten steel enables his customers to operate equipment in temperatures as low as –45° F.

Extreme cold causes carbon steel to lose some of its toughness, whereas, under identical conditions, steels containing nickel retain a high degree of toughness. In addition to resisting shock, nickel alloyed steels permit cutting bulk and dead weight without sacrificing strength or safety. For example, in some applications the weight of Tri-Ten steel parts was cut 25 percent, despite a 50 percent increase in working stresses.

To assure dependable, trouble-free performance, specify nickel-containing alloys in *your* equipment. We invite inquiries regarding specific applications.

At the present time, nickel is available for end uses in defense and defense supporting industries. The remainder of the supply is available for some civilian applications and governmental stockpiling.

EMBLEM OF INCO SERVICE

THE INTERNATIONAL NICKEL COMPANY, INC. 67 WALL STREET NEW YORK 5, N. Y.

International Nickel Co. of Canada Ltd. is undertaking production of byproduct iron ore from nickel ores in the Sudbury District of Ontario. Inco plans immediate construction of a $16 million plant in the Copper Cliff area which will treat 1000 tons of pyrrhotite a day, yielding iron ore in addition to nickel. Iron ore is expected to contain at least 65 pct iron and less than 2 pct silica.

Bolivia signed a contract to sell 10,000 tons of tin concentrates to the U. S. Shipments were scheduled to start late in September to a Texas refinery. Payment will be based on New York market prices taken on a monthly average. The Bolivian Mining Corp. announced that there were 5000 tons of concentrates at Pacific ports ready for shipment.

Holdings of the Brunswick Mining & Smelting Co. and the joint operation of Anacon Lead Mines and Leadridge Mining Co. in the Bathhurst, N. B., field have been consolidated. Leadridge Mining Co. is a wholly-owned Canadian subsidiary of St. Joseph Lead Co. Production plans for the big base metal orebodies have not been finalized.

Newmont Mining Co. purchased the one-third share in Resurrection Mining Co. owned by Hecla Mining Co. Previously, U. S. Smelting Refining & Mining Co. had sold its one-third interest in the Leadville, Colo., operation to Newmont, now in control of the entire property.

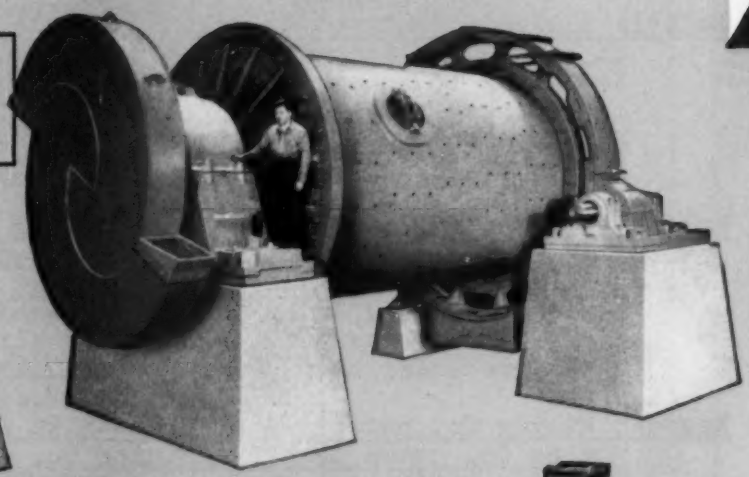
Melbourne University scientists are engaged in research aimed at finding improved methods for reclaiming titanium from Australia's hundreds of miles of sandy beaches. Titanium and zirconium concentrates are already being recovered along a 50-mile stretch of southeastern Queensland coast. Spirals are used to treat the black sands.

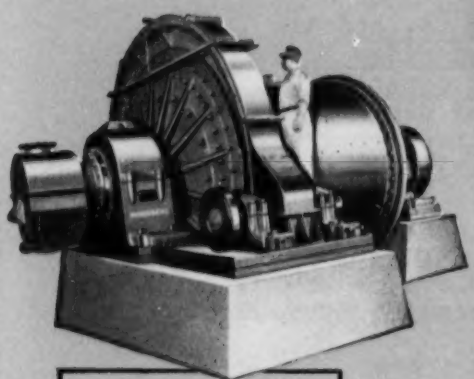
Consolidated Mining & Smelting Co. of Canada Ltd. termed the new electric induction furnace for melting zinc sheets, now in operation at its Trail, B. C., plant, the largest of its kind in the world. Designed and constructed by Societa Italiana Costruzioni Elettrotermiche, Milan, it is rated at about 300 tons of zinc per day.

Sherritt Gordon's pilot plant at Ottawa was turned over in initial tests to determine the economy of treating copper concentrates at Fort Saskatchewan, Alberta, where a large nickel treatment plant is nearing end of construction. Preliminary tests, it was reported, have indicated that treatment of copper concentrates by the Forward process of Sherritt Gordon is possible and may offer substantial savings over standard practices. MINING ENGINEERING, June, 1953 covered the Sherritt Gordon operation in the first detailed article on the process to be published in the U. S.

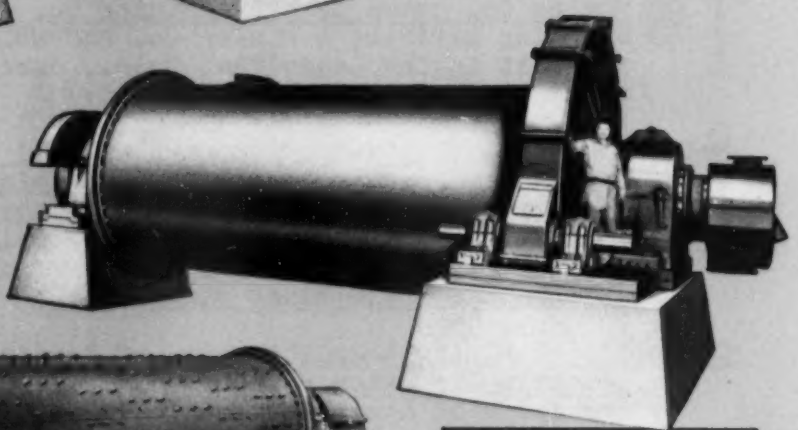
NORDBERG GRINDING MILLS

NORDBERG 10'-8" x 17' Wet Grinding Mill showing scoop feeder.





NORDBERG 9'-6" diameter x 11' long Dry Grinding Mill.



NORDBERG 9'-6" x 24' Wet Grinding Pebble Mill for non-ferrous milling.

NORDBERG 8'-6" x 36' two-compartment clinker Grinding Mill manufactured for a large cement plant.

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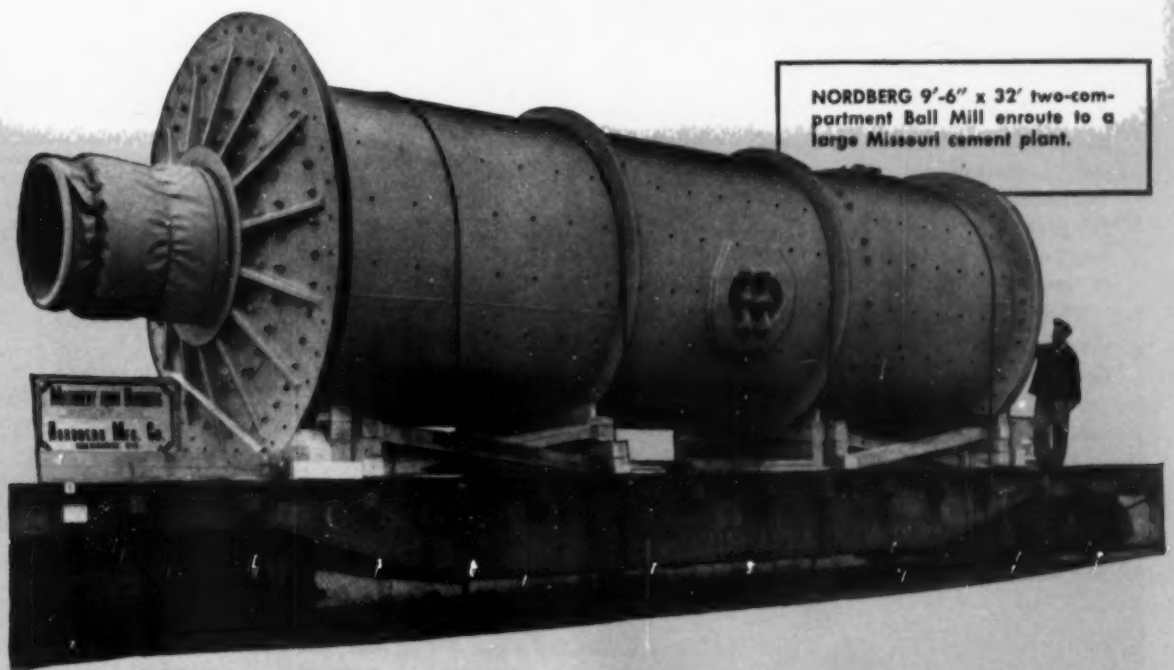
This is evidenced by the universal acceptance which Nordberg Machinery has received by most of the Mining and Mineral processing industries throughout the world.

Typical of this heavy machinery is the complete line of Nordberg Grinding Mills available for the manufacture of Cement, the extraction of Metals from Ores, and in the processing of Non-Metallic Minerals, which require comminution to fine sizes.

Nordberg manufactures Ball, Tube, Rod and Compartment Mills, of grate, overflow and peripheral discharge types for wet or dry grinding service, in sizes ranging to 13′-0″ diameter, and in lengths to 40′-0″

Competent engineering services are available for new processing plant design or modernization of existing plants.

For large or small operations, you can depend on Nordberg Machinery to deliver maximum output at lowest possible cost.



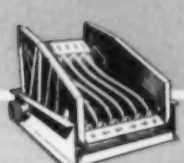
NORDBERG 9′-6″ x 32′ two-compartment Ball Mill enroute to a large Missouri cement plant.

SYMONS GYRATORY CRUSHERS

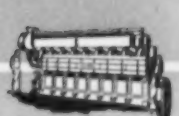
SYMONS CONE CRUSHERS

ROTARY KILNS and COOLERS

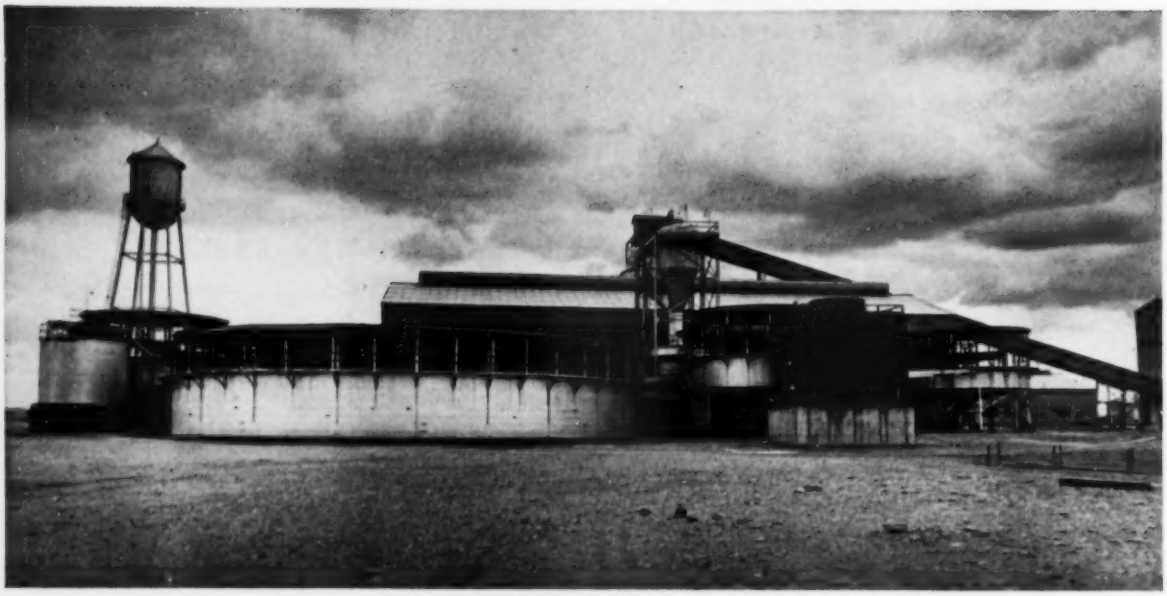
SYMONS "V" SCREENS



SYMONS VIBRATING GRIZZLIES and SCREENS



DIESEL ENGINES — 10 to over 10,000 hp.



Overall view of the plant with a 120′ dia. Dorr Thickener in the left foreground handling flotation tailings. A 40′ dia. Dorr effluent Thickener and 40′ dia. Dorr Hydroseparator are at the right. All three units are covered to prevent temperature increases resulting from the sun's rays.

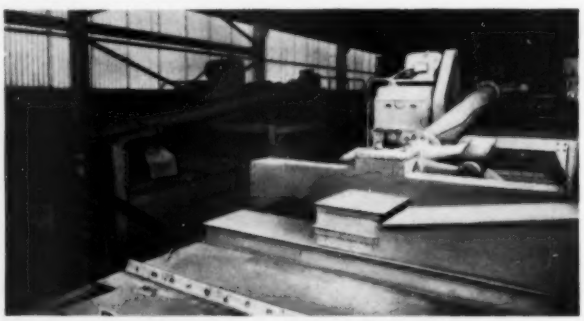
NEW MEXICO POTASH PRODUCER

Combines Chemical and Metallurgical Techniques at New Plant . . .

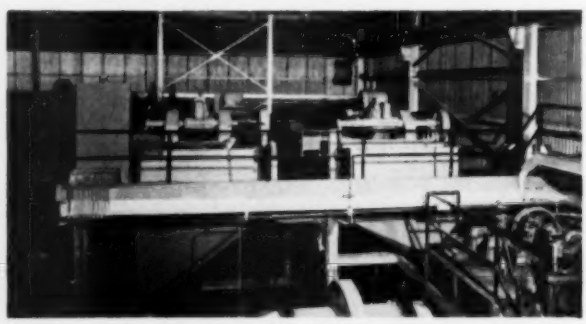
A new U. S. producer for the quality potash market, Duval Sulphur & Potash Company recently started up this modern plant near Carlsbad, New Mexico. A perfect example of the growing convergence of chemical and metallurgical techniques, the plant makes skillful use of the latest developments in both fields.

The flowsheet includes crushing, grinding, classification, hydroseparation, flotation, thickening, centrifuging, crystallization and drying.

At Duval, as throughout the chemical and metallurgical industries, Dorr equipment and techniques play a vital part in wet processing flowsheets. For more information on the scope of the Dorr technical service, write for bulletin No. 7002. The Dorr Company, Stamford, Conn. *In Canada:* 26 St. Clair Avenue, E., Toronto 5.



Closeup of two Duplex Dorr HX Classifiers. The Primary at right handles raw crushed salt which has been pulped with return brine. Sands go to a rod mill and fines to the Hydroseparator.



General view of the milling section. Rod mill discharge goes to the Secondary Classifier at the right. Sands are discharged to conditioners and fines overflow to the Hydroseparator.

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Trackless Equipment, Bigger Shovels Set Trend

Trackless underground equipment and larger power shovels are helping to fight costs in U. S. mines, according to statements made at the American Mining Congress Convention, September 21 to 24, at Seattle. Conversion to trackless equipment slashed mining costs per ton of ore by almost 50 pct at Pend Oreille mine, a Washington lead-zinc producer. According to L. M. Kinney, general superintendent of the Pend Oreille Mines & Metals Co., production of ore for a selective flotation mill has increased from 800 to 1600 tons per day since introduction of trackless equipment.

Also contributing to greater production has been a change in drilling methods. A two-man crew in a stope using a bar or tripod-mounted drill averaged about 100 ft of drilling per shift. Now the crew employs trackless drill jumbos for an average 400 ft of drilling per shift.

Using the slusher method, mining costs averaged $2.15 per ton, Kinney said, but trackless equipment has reduced this figure to $1.14 per ton. Trackless equipment has made the mine "a profitable one instead of a losing proposition under present low metal prices," he said.

Adrien F. Busick, Jr., vice president and chief engineer of the Marion Power Shovel Co., feels that the trend toward larger power shovels with dipper capacities up to 11 cu yd continues unabated. He said the trend resulted from steadily mounting labor costs; deeper overburden covering minerals in many areas; use of lower grade ores in many areas; and intensive development of ever-larger trucks.

Commenting on the choice of excavators, R. M. Dickey, a Bucyrus-Erie Co., sales manager, said that major consideration should be given to interest, taxes, and insurance on the all-in cost of the installed machine. Repairs, maintenance, and supplies for the machine should also be of prime interest.

He stressed the desirability of smaller machines for certain tasks. Mr. Dickey noted that pit layout may be impractical for providing the larger number of working places needed for smaller shovels to meet desired output. Thus, more consideration would have to be given to shovel size rather than to direct operating costs.

Where mobility, ease of movement, and short-term rapid output are desirable in loading of selected ore grades, smaller excavators in greater numbers may be more suitable.

New Division Takes Over DMPA Assignment

Elimination of red tape is given as the reason for establishment of a Materials Division in General Services Administration to take over duties formerly handled by DMPA.

The new division is part of GSA's Emergency Procurement Service, which handles stockpile procurement, transportation, and storage. GSA Administrator Mansure said, "I am locating our new responsibilities in our regular stockpile organization because of the intimate relationship between the two programs." Chief of the Materials Div. will be Irving Gumbel.

Mansure commented that: "We expect to continue a vigorous program to increase the potential of scarce metals and minerals. There will be no let-up to reach the goals set by the Office of Defense Mobilization, which is charged with the general direction of mobilization activities and the national stockpile.

"I urge everyone who has dealt with DMPA, and others who may wish to do so, to recognize that the change is one of internal organization—cutting out unnecessary administrative machinery—and that we are still going forward with the metals program."

Battelle Investigates Howe Sound Cobalt Plant Output Delay

Howe Sound Co. has engaged Battelle Memorial Institute to make an on-the-job study of the firm's cobalt refinery at Garfield, Utah, in an attempt to clear up frequent delays which "have prevented a continuity of production."

Howe Sound reports that it has found difficulty in adapting new, specifically designed equipment to meet conditions of corrosion and abrasion inherent in the chemical refining process.

Cobalt up to government specifications has been produced at the refinery for the past several months. However, because of recurrent delays, production is not up to a commercial level, according to a Howe Sound official.

Battelle, with offices at Columbus, Ohio, has considerable prior knowledge of certain phases of the process used at Garfield, and a report of its study, which began in June, is expected either late in September or early October.

Howe Sound reiterates its belief in the soundness of the process and that current problems will be solved. However, pending the report, the company makes no prediction of when full commercial production will begin.

The cobalt plant at Garfield is the first commercial size plant in production to use the chemical refining process for recovery of base metals. Other plants are in various stages of construction and pilot experimentation in the U. S. and Canada.

At Coleman Collieries Ltd., 7400 ft up Tent mountain in the Canadian Rockies, a Caterpillar powered Bucyrus-Erie shovel removes overburden at what is claimed to be the highest coal mine in the world.

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Climax Residents Tune in on Video

Television is coming to Climax, Colo. Sometime this fall or early winter, residents of the Climax Molybdenum Mining Co. camp will sit in their living rooms and watch a picture beamed from Denver, some 73 miles distant. Climax winters are cold and snowfall heavy, spelling out long evenings at home. Thus, when Bob Ver Steeg and four other company employees climbed through heavy snow to the summit of McNamee Peak and received a definite video signal from Denver, it opened a new world to Climax people. By June 17, the third trip to the peak, an actual picture had been received.

Various television companies were called in to verify the signal and advise on equipment for the most satisfactory picture. An antenna mast is installed in concrete and will carry four heavy Yagi type antennas, one tuned for each of four channels. There will also be a dipole for FM reception. At last report only an amplifier shack remained to be installed at the summit, before 12,000 ft of coaxial cable is stretched down the mountain to the town.

From a central point in Climax the cable will fan out to homes over existing power poles. Amplifiers will be used to keep the signal at the same strength as found on top of the mountain.

One of the "different" features of the installation is that residents of the town will not be charged for television reception service. However, there will be a small hookup fee when sets are connected to the community antenna. "This charge will only be for the service rendered at that time," emphasized Resident Manager Frank Coolbaugh. The company is also making a careful study of television receivers to determine which sets are most suited for use at Climax.

Floyd Miller, Collier Electric Co. lineman, struggles to load 300 lb of TV equipment on the guy wire cable of the aerial tram. Two sections of antenna are lashed to a stretcher basket fitted with pulleys.

One of the engineering feats accomplished by the group of men who were responsbile for erection of the antenna was construction of an aerial tramway for transport of equipment up the mountain. Equipment had to be moved some 2400 ft to the top from a working camp. Despite careful planning, however, the cable tended to sag. Men were forced to fight rocky terrain every time a load went up the slope.

June 17 was the big day. A true television picture was seen on top of Mt. McNamee after a crew of 18 lugged some 1000 lb of equipment to the summit. Picture was received with a 30-ft antenna rigged to a set driven by a gasoline motor driven generator.

Kerr-McGee to Build Uranium Facilities

The U. S. Atomic Energy Commission and Kerr-McGee Oil Industries Inc., signed a contract for construction and operation of a uranium ore processing plant at Shiprock, N. Mex., according to Sheldon P. Wimpfen, manager of the AEC Grand Junction Operations Office.

The Commission has operated a sampling plant at Shiprock for the past two years. Ores produced in the northern part of the Navajo Indian Reservation have been stockpiled, awaiting construction of a processing plant.

The Navajo Uranium Div. of Kerr-McGee will construct and operate the processing plant. Clyde Osborn, manager of operations, is in charge of mining and milling activities, and G. R. Kennedy is manager of exploration.

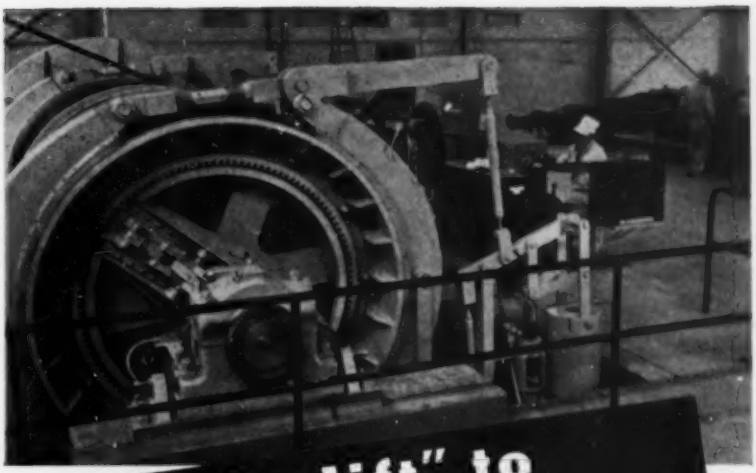
Milling of uranium ore is expected to start sometime in 1954. Execution of this contract will make a total of ten uranium processing plants either operating or under construction on the Colorado Plateau.

AEC to Post Ground Reconnaissance Results

The Atomic Energy Commission announced through its Grand Junction Operations Office, Grand Junction, Colo., that location of surface areas with unusual radioactivity found by ground geological investigation will be posted.

Previously, only areas of high radioactivity found by airborne detection were included in public releases. Notices will be posted on the 15th of each month, or the next workday if the 15th falls on a Saturday, Sunday, or holiday, at the following offices:

New York Raw Materials Office, AEC, New York City; Denver Exploration Branch, AEC, Denver Federal Center, Denver; Hot Springs Suboffice, AEC, Hot Springs, S. Dak.; Bureau of Mines Office, Rapid City, S. Dak.; U. S. Geological Survey Office, Custer, S. Dak.; Grand Junction Operations Office, AEC, Grand Junction, Colo.; Grants Suboffice, AEC, Grants, N. Mex.; U. S. Atomic Energy Commission Buying Station, Shiprock, N. Mex.; Salt Lake Exploration Branch, AEC, Salt Lake City; Richfield Suboffice, AEC, Richfield, Utah; U. S. Atomic Energy Commission Buying Station, Monticello, Utah; Butte Suboffice, AEC, Butte, Mont.; U. S. Geological Survey Office, Worland, Wyo.; U. S. Geological Survey Office, Casper, Wyo.; Albuquerque Suboffice, AEC, Albuquerque, N. Mex.; Douglas Suboffice, AEC, Douglas, Wyo.; Phoenix Suboffice, AEC, Phoenix, Ariz.; and Ishpeming Suboffice, AEC, Ishpeming, Mich.



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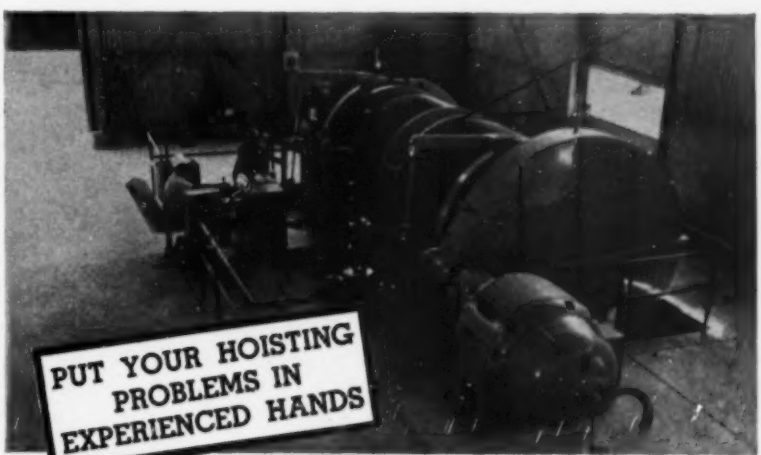
Typical of hoists designed and manufactured by Stearns-Roger *for specific requirements* is this twin drum, 22,000 lb. electric hoist installed in a large Western mine.

Each drum is cast Meehanite®...84" diameter with 78" face, winding 1,200 feet of 1¾" cable. One drum is fixed and the other clutched. Brakes are set by gravity and released hydraulically. Drums are fitted on 10" shafts with three self-aligning bearings, with two such bearings serving the 6" pinion shaft.

Rating is 22,000 lbs. rope pull at 1,000 ft. per minute rope speed.



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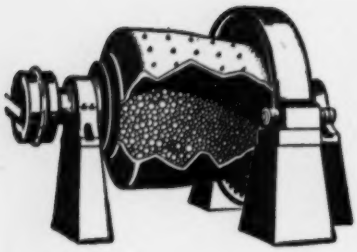

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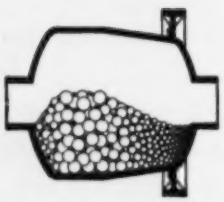
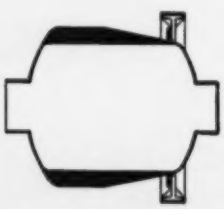
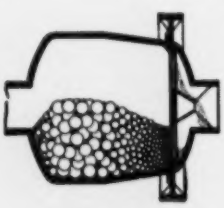
TRICONE
MILLS



CORRECT BALL SEGREGATION



GREATER GRINDING VOLUME



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Color Air Photos Provide New Data

Use of color in aerial photography is gaining importance in discovering hidden ore deposits, Philip A. Laylander, Nevada aerogeologist, told delegates to the recent convention of the American Mining Congress at Seattle.

Laylander described the use of stereoscopic color photography to determine geologic structures. The photographs provided a three-dimensional view of the finer details of tones, textures, and patterns of structure-alteration — ore relationships that cannot be obtained by ground study alone, he maintained.

Study of several western mining districts showed that many geologic maps and reports contain descriptive and analytical errors, Laylander said. Mistakes included the common error, according to the aerogeologist, of failing to recognize regional thrusts which controlled emplacement of tertiary intrusives.

Aerial color photographs "often indicate directly ore situations to the point where mining and drilling is indicated," Laylander said.

Consulting Firm Designs Orinoco Co. Laboratory

Sam Tour & Co., New York City research and consulting firm, designed a completely equipped laboratory to handle analytical and assay work for the Orinoco Mining Co. at Cerro Bolivar, Venezuela.

The consultants designed the laboratory from personnel to pulverizers. As planned, the laboratory could operate six months without additional supplies. The mine, located some 55 miles south of the Orinoco River, is too remote for frequent supply delivery.

Step by step procedures were developed for all analytical work to be handled by the laboratory. Analyses include: iron, silica, alumina, phosphorous, manganese, titanium, moisture, and organic materials.

Nearly 200 samples will be analyzed during two 8-hr shifts. Time limits have been placed on some key analyses so that outgoing ore trains can be directed en route to the proper collecting points. Some of the standard analytical procedures were revised to reduce work time to specified limits.

Six Ball Mills For White Pine Copper

Six ball mills, each 12½x13 ft inside the liners, will be employed in the primary grinding section of the White Pine Copper Co., White Pine, Mich.

Weighing approximately 300 tons complete with balls and liners, the mills will be powered by General Electric 1500-hp, 150 rmp synchronous ball mill motors. The mills will be in closed circuit with spiral classifiers.

Three GE 500-hp synchronous motors will drive the tube mills in the secondary grinding section. Included in the more than $750,000 worth of equipment supplied to the White Pine project by GE are 250 flotation-cell induction motors, metal-clad switchgear for the power house, conveyor drive motors, 16 portable underground ac substations, and a 45-ton diesel electric locomotive.

The program at White Pine is expected to cost $57 million. Production is scheduled to start late in 1954.

The new Airslide car is designed for bulk shipping of dry, powdered, or granular commodities. The car handles high or low density materials equally well, according to General American Transportation Corp. and the Fuller Co. It unloads by gravity feed, utilization of a mechanical conveyor, or pneumatic conveyor.

Pennsylvania in Mine Dewatering Experiment

The Commonwealth of Pennsylvania is undertaking an experimental pumping operation in an attempt to save anthracite coal mines from flooding. Funds supplied by the state will serve as stopgap until a major drainage project is constructed.

A pump with a 20,000 gpm capacity will be installed at the Twin Shaft in Duryea. Another pump with a 10,000 gpm capacity is to be installed at the Exeter Shaft near Exeter. Once owned by Lehigh Valley Coal Co., the mines are now in independent hands.

Funds from the Commonwealth will be employed as a first-aid measure until Congress supplies money for the projected drainage system to empty into the Susquehanna River at Conowingo, Md. Cost of pumps is estimated at about $300,000. If the experiment works it is reported that it will be extended to other areas in the state. Some coal is still mined in the upper levels of Twin Shaft. However, Exeter Shaft has been abandoned.

The General Assembly of Pennsylvania has appropriated $1.5 million for the experiment. Congress is expected to vote funds sometime in the future for the tunnel project which will take five years to complete.

Germanium, Uranium Found on Cape Breton

Germanium has been reported on Cape Breton Island, Nova Scotia, along with copper, nickel, and three other metals. The report states that signs of uranium were also found over a wide area in northern Cape Breton.

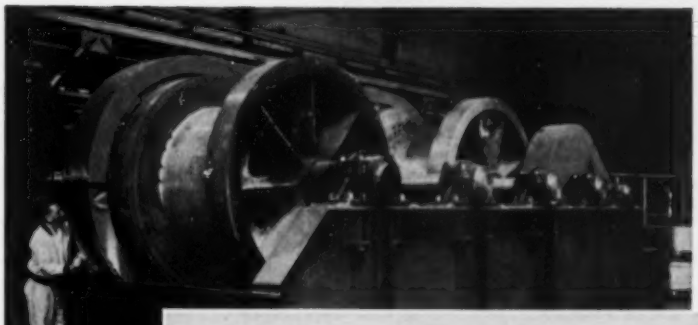
Minerals found were reported to be in commercial amounts, while uranium "may be of great value." A million-acre tract has been leased to Mineral Exploration Corp. of Toronto.

3 Gold Mines Produce South African Uranium

South African Minister of Mines Johannes Viljoen announced that three gold mines are producing uranium and that a total of 22 mines are expected to be in production by 1955.

The South African Government recently invited a U. S. Joint Congressional Committee to pay a courtesy visit to various plants on the Rand and in Orange Free State. South Africa borrowed some $112 million from the U. S. to build a uranium plant.

YUBA-BUILT HOISTS
Lift 1000-Ton Loads



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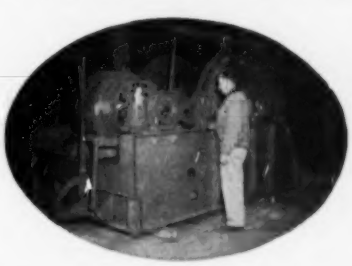
This YUBA-built double-drum dredge hoist is designed to handle 194-foot digging ladder and is capable of lifting 800 to 1000 tons. Single-drum hoist (bottom) built by YUBA for tramway carries 5200 feet of wire rope. We're old hands at building such machinery. Every bucket ladder dredge Yuba has built combines many special "machines"—hoists, conveyors, drives, screening and metal saving equipment, winches, pumps —all designed and built by Yuba.

Experience and Facilities

YUBA's manufacturing experience includes grooved drums for large gate hoists, barges, steel and chemical plant equipment, construction rigs, clamshell buckets, ordnance, other custom work, and structural steel production. Parts for field assembly are fitted and match-marked before shipment to insure accuracy.

Our facilities include shops for carbon and alloy steel forging, heat treating, steel fabricating, welding, pattern work, iron and bronze castings, and machine shop capable of handling single large, heavy pieces. We have long experience in properly preparing and documenting export shipments.



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65

"Excellent" *says J. Robert*

7 Tournapulls help strip 7,500,000 tons overburden

Contract stripping over 7,500,000 yards of overburden near Mt. Carmel, Pa., to uncover anthracite deposits, J. Robert Bazley, Pottsville, brought in 7 electric-control Tournapulls. Of these high-speed units, 5 were 14-yd. C Tournapulls . . . the other 2 were new 27½-yd. "A's".

Big "A's" complete 5000′ cycle every 8¾ minutes

Loading tough hardpan, the "A's" got heaped loads of the high-void clay and rock in about a minute. The day pictures were taken, a quick freeze had left the area cut up with frozen ruts and ridges, which slowed both haul and return. Yet, each of the "A's" regularly made a 5000′ cycle every 8¾ minutes, completed 48 trips per 7-hour shift. Working 2 shifts per day, 5 days per week, combined production for the 2 big units has been 233 pay yds. per hour, according to contractor's records. "This LeTourneau-Westinghouse equipment is satisfactory," says Owner J. Robert Bazley. "It has done an excellent job."

General Superintendent Harry H. Hughes declares, "Our LeTourneau rigs have given satisfactory service on this and other jobs."

Prime-movers also operate rear-dumps

In addition to scraper work, the Bazley organization uses its electric-control LeTourneau-Westinghouse prime-movers with rear-dump Tournarocker trailing units on rock work. These extra rear-dump bodies cost only about 25% of the cost of original unit, and are readily interchangeable with the scrapers. Prime-movers thus do *double duty* . . . working steadily month after month where needed most. As James R. Bazley, company Vice President, puts it, "Interchangeability of Scrapers and 'Rockers permits us to use our Tournapull prime-movers the year

Stripping in pit began in April 1947, is expected to continue through 1955.

Tournapull returns to stripping pit down 8% grade. Top speed for 27½-yd. rig is 36.5 mph. Average speed on this 5000' cycle was 12 mph. Same "A" prime-mover interchanges on a 35 or 50-ton Rear-Dump or Bottom-Dump. "C" prime-mover powers 19-ton Scraper, Rear-Dump, and Bottom-Dump.

Pushed by a 167 hp tractor, big 27½-yd. A Tournapull picks up heaped load of tough hardpan, in spite of 38% voids in rocky material. In background, second Tournapull turns 90° to move into loading position.

Bazley

around. By switching the trailing units, we adapt our equipment to the prevailing conditions for the most economical stripping."

Get facts for your work

Whenever you have dirt, shovel-rock, sand, coal, or other bulk materials to move, it will pay you to check the fast, low-cost work-capacity of LeTourneau-Westinghouse equipment. Ask your Distributor for all the facts. Have him arrange a demonstration on your job. There's no obligation.

Tournapull, Tournarocker—Trademark Reg. U. S. Pat. Off. PAP-381-CM

LeTourneau-Westinghouse Company

PEORIA, ILLINOIS

TAX TIPS — TOURIST MINERS — SAUGUS IRON WORKS

WILLIAM J. Casey and J. K. Lasser, men who have been telling the taxpayer how to survive with honor—and a little cash—under the present U. S. tax structure, have completed a report on how business may hope to hold its own and even to grow under today's conditions. They give dozens of examples of how leading firms have moved in recent years and are moving today to protect operations.

They show, for example, how Henry Kaiser, and Robert Young used losses to acquire profitable businesses; how real estate operator William Zeckendorf gave himself a liquid position and substantial tax protection by merging his real estate operation into an unsuccessful investment trust; how Ford Motor Co. used a distributorship to develop a capital position for its executives; and how many other firms placed themselves in a better tax position.

They point out that one of the major strategies in building a business under today's tax structure is to spend tax-deductible dollars in a way which will build up future capital values. Advertising can work as a tax shelter, according to the authors. It can underpin plant capacity value and dividend yield.

Research is also evaluated. The experts show how Minnesota Mining & Manufacturing Co. has translated research expenditures into a sensational growth in capital values and how such companies as National Cash Register and Remington Rand have acquired research value by tax-free mergers, by offering capital gain deals, and by tax deductible expenditures.

What the authors appear to be advocating is a more aggressive attitude on the part of industrialists who have been taking a beating over the last few years. They seem to think that taxes as they stand can be lived with, providing business takes advantage of the avenues of endeavor left open by the law. In no way do they advocate circumventing the law. Rather, they offer a multitude of ways adopted by companies that want to survive and to grow. The authors feel that the "most charitable thing that can be said of our tax structure is that it is badly in need of an overhauling which will allow all of us to spend more of our thought, time and effort in satisfying each others needs and less in arranging our affairs to protect our families and our businesses from crippling tax burdens."

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IT would appear that defunct mining operations are in reality lying fallow until some enterprising promoter goes to work on redeveloping the property along other lines. Mines are being used as underground medical centers whose supernatural powers can cure anything from a cold to arthritis, as atomic bomb protection vaults for valuable papers, and as tourist attractions.

The latest on the scene appears to be a couple operating abandoned mines near Idaho Springs, Colo. Not long ago, K. D. Kaasch and his wife leased the Gold Star and the Red Elephant mines. During its production life the Gold Star is said to have produced millions in gold, silver, and lead. Today, it's still producing gold—from tourists who want to see a *real* mine.

The Fat Chance Mining Co., as it is called now, has four sites. Two of them are tunnel mines, strictly for guided tours. Two others, however, are placer type mines. Tourists can slosh water around in a pan or a sluice box—and even find a few flakes of gold. Pay dirt is hauled in to guarantee each customer will find something at the bottom of his pan. It consists of gold-bearing sand and whatever the customer finds is placed in a little bottle for him. A few hardy souls even borrow pans and take off by themselves—but thus far no one has made a strike. As someone said, "There's still gold in *them* hills."

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RESTORATION of the Saugus, Mass., iron works, which many consider to be the cradle of the American iron and steel industry, is making progress. Miss M. Louise Hawkes, a smallish, white-haired lady, is the one most responsible for this preservation of a piece of colonial history. The iron master's house was restored in 1915, and a group of Ford Trade School graduates bought it in 1942. They intended to move it to Dearborn, Mich., site of Ford's Greenfield Village. Miss Hawkes, whose Saugus ancestry dates to 1630, fought the idea and with the intervention of Mr. Ford, the house remained at Saugus.

It now houses the headquarters of the First Iron Works Assn., guiding force behind the restoration. Bulldozers, cranes, shovels, and other modern machinery are helping to rebuild the works as it was when established in 1646. Restoration will be completed in about one year, culminating a ten-year dream. The American Iron and Steel Institute is providing financial help. Professor E. N. Hartley of MIT is doing the historical research necessary for authentic restoration, while Roland Wells Robbin has been the archaeologist. Mr. Wells also discovered the site of Thoreau's cabin on Walden Pond.

The hammer head of the original forge and the waterwheel that powered the blast furnace bellows have been found with the aid of a mine detector. John Winthrop, Jr., son of the first governor of the Massachusetts Bay Colony, was responsible for the establishment of the original iron works. He and Richard Leader, an English merchant, formed the Company of Undertakers for the Iron Workes in New England. Originally, the Saugus River settlement was called Hammersmith. The furnace is said to have produced for about 25 years, turning out cast and wrought iron from ore found in nearby bogs. Charcoal, ore, and fluxing agents were fed into the top of the furnace. Some rock ore was also used. Farmers cut enormous quantities of wood

from local forests for charcoal. In 1646 Leader purchased some defective cannon and thus became the first American iron master to use old metal as scrap.

The iron works was of inestimable economic benefit to the colonies. It made many items that were previously done without, or substituted for in a rather crude manner. It turned out gears, pots, spikes, cannon balls, weights, and other necessities which took months to come from England.

Carpenters are nailing seventeenth century clapboards in place. The outlines of the old iron works are taking shape. Behind its restoration lies a spirit that is essentially American and the old Saugus iron works will serve to remind many visitors that here was the beginning of a mighty industry.

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DOMESTIC tungsten miners have been faced with an economic pattern of feast and famine for the past 36 years. During the mad scramble of World War I tungsten zoomed to $100 per unit of WO_3. When surplus tungsten hit the market at the end of hostilities the price collapsed to $1.75 per unit for a time. U. S. tungsten mines closed and stayed that way until the late 1920's. World War II saw a healthier situation evolve with the Government paying $24 per unit for tungsten from old mines and $30 for tungsten from new operations. By 1949, however, tungsten fell to $19 per unit and almost every mine in the country, with the exception of the one near Henderson, ceased activity.

The Government once again came to the rescue of the industry when it decided to pay $27 per unit in 1950. The support was to have lasted for only one year—but with the outbreak of hostilities in Korea, tungsten skyrocketed to a high of more than $86 per unit in March of 1951. A ceiling of $65 was imposed along with a buying program which is still in force. The Government program was to have lasted five years, but only 283,495 units were delivered by last June. Today, the tungsten supply situation is one of the most important problems in the U. S. stockpile program. China produces about one-third of the world's annual output and it can be safely assumed that Russia is now getting almost all of that.

A new law signed by President Eisenhower guarantees tungsten miners a market for the major part of their production until 1958 at a floor price of $63 per unit. Foreign tungsten is selling for about $43 per unit, to which must be added an import tax of $7.91. During 1952 the U. S. imported about 17,864 tons of tungsten concentrates. Most of it came from Korea, Bolivia, Portugal, Spain, Thailand, Brazil, Burma, and Australia. The greater part of the imported material went into stockpile. Domestic production reached 7,168 tons—the highest since 1944—against a consumption of 9,072 tons.

Obviously, the U. S. would be in serious trouble should war come. With most of U. S. tungsten sources a great distance away and with submarine warfare developed to almost a mathematical skill, military needs would be critical.

Thus, the move of ten of the nation's largest tungsten producers to form the Tungsten Institute appears to be of concrete value. The Tungsten Institute aims at a program of research and development to encourage growth of tungsten mining in this country. They want to make the U. S. self-sufficient in the event of national crisis. While the U. S. produces 20 pct of the Free World's tungsten, the Institute intends to promote the development of known large deposits and the exploration of undeveloped prospects.

But tungsten in war is not the only objective of this group. They point out through W. Lunsford Long, the Institute's president, that tungsten has innumerable peacetime uses—and the list is growing. He envisages tungsten as a powerful force in building a better world: ". . . the potential applications of tungsten in industry have scarcely been touched yet, and we're looking forward also to the day when the stockpiling program ends and domestic tungsten must hold its own in the world market."

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NO comics, sports, battlefront reports, or sensational news, and yet one British newspaper boasts a circulation of more than 200,000 readers. The paper is called *Target*, and has a diversified readership which includes technical college students, factory managers, shop stewards, and trade union officials. These people don't have to pay for it except through taxes, but they do have to renew their requests in writing from time to time. *Target*, published by the British Government, is concerned with only one thing—cutting waste of time and materials in United Kingdom plants.

It uses no tricks of journalism to spotlight accidents and production bottlenecks. One device it does use is that of the picture layout. The photographs are used to illustrate how science can help produce more and better goods, how a new machine will increase production, or how a better method of quality control can be achieved.

Language is straightforward, designed to communicate ideas in their simplest terms. Where new methods and techniques have succeeded full credit is given to those responsible. Hundreds of histories have been published which show how plants solved definite problems which plagued other members of industry. Labor-management problems have been given particular attention.

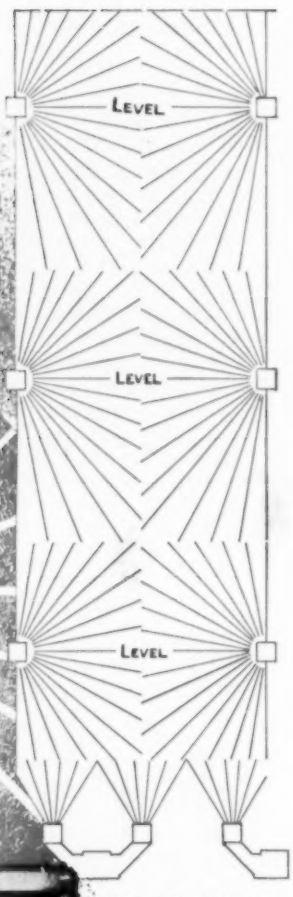
An editorial in *Target* put it this way:

"Any firm which finds a better, cheaper, more efficient way of producing goods of high quality can help the whole country by sharing its knowledge and experience."

Some of the largest and most modern British companies have taken ideas from the newspaper, while others have used reports as starting points for idea-producing sessions.

GARDNER-DENVER is first again!

now it's the "Blast Hole" drill designed especially for percussion deep hole drilling



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A Natural Partner — with Carbide Bits and Superior Gardner-Denver Alloy Sectional Rods and Couplings — for Low Cost Blast Hole Drilling.

Permits a 4' change in a 7' drift—saves time and development work.

Superior chuck construction—1½" x 3 13/16" shank rod locked in chuck—results in longer life for hammer, tube, chuck, shank rod and ring seal.

Locked-in shank rod prevents loss of rod string on down holes.

Heavy-duty centralizer supports rod string during rod changes—simplifies up-hole drilling.

Reverse rotation and locked-in shank rod enable operator to uncouple rods quickly and easily—using power from the drill and only one wrench.

Delivers high-pressure water or air through chromium plated tube and Ring Seal Shank.

Lightweight aluminum alloy guide shell—reversible for extra life. Movable cone.

Write today for complete details.

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Sulphur

Molten Sulphur flowing into the storage vat

Thousands of tons mined daily, but where does it all go?

Look around you in any grocery store and what do you see — canned goods of all kinds! Soups, vegetables, fruits, berries! Believe it or not, you're looking at merchandise that consumed a lot of Sulphur in the making.

Tin cans are made of tin plate. Tin plate is made of sheet steel. Sheet steel is made with the help of sulphuric acid—pickling, as they call it, the process that removes scale preparatory to plating. In 1951, the sheet division of our great iron and steel industry is estimated to have consumed 140,000 long tons of Sulphur in the form of sulphuric acid. That in itself makes quite a dent in our supplies of Sulphur. Add to this almost as much more for treating wire rod, plate, strip, bars, etc., and you can see that to make finished steel, regardless of form, the iron and steel industry must use lots of Sulphur in the form of sulphuric acid.

Right here is an excellent example of the interdependence of all of our industries. To produce steel requires a lot of Sulphur. To produce Sulphur and other mined products requires a lot of steel. This interdependence of industries is one of the country's sources of strength.

Texas Gulf Sulphur Co.

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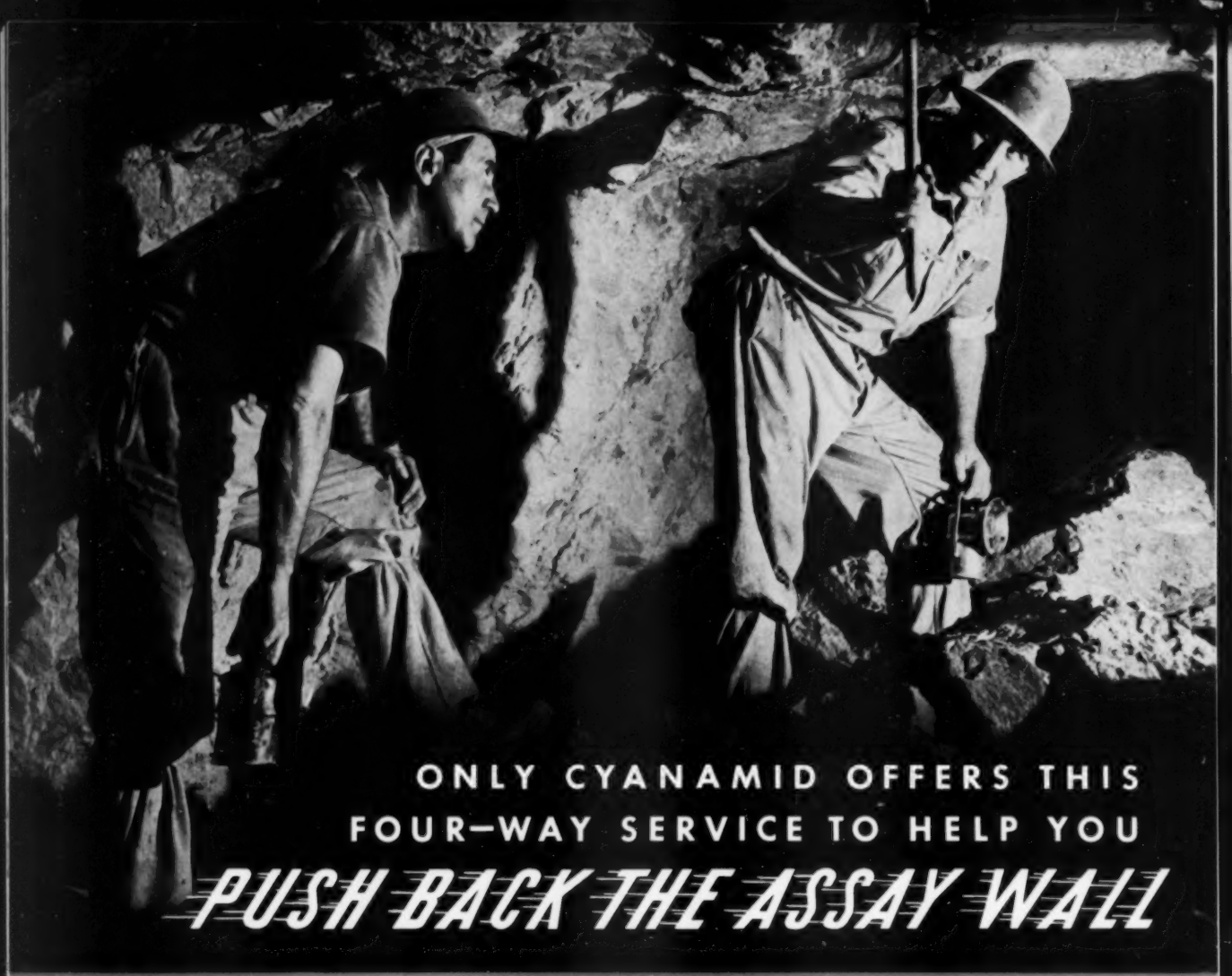

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Cyanamid alone offers a most complete line of chemicals, both for cyanidation and flotation. With Cyanamid's long line of reagents you never need compromise one iota on the exact reagent combination for highest recovery.

In promotors, for example, Cyanamid offers a choice of twenty-two Aerofloat* and Aero** Promotors plus a complete line of Sodium and Potassium Xanthates. Also available are a variety of Aerofroth† frothers, Pine Oil and Cresylic Acid.

Year by year, Cyanamid has developed such noteable special purpose reagents as the 500 Series Depressants, the 400 Series, and the 800 Series Promotors for oxidized metallic and non-metallic ores. New methods have also been developed which have wide application for thickening and filtration of ore and other pulps. Other reagents are being aggressively researched.

2. FIELD ENGINEERS

Backed by the technical resources of Cyanamid Mineral Dressing Laboratory and kept constantly informed of progress in every mining field, Cyanamid Field Engineers are literally "walking encyclopedias" of current beneficiation practice.

But, by the same token, they are eminently practical mill men who can give you in-the-mill shirt-sleeves service on perplexing treatment problems. Their primary job is to help you work out the most efficient reagent combination and flow scheme.

3. DELIVERY SERVICE

To implement our application service, we maintain substantial warehouse stocks of metallurgical chemicals for cyanidation and flotation at strategic points. Frequently, mills can achieve substantial savings through lower-cost combination shipments and better balanced reagent inventories.

Through periods of world upheaval spanning thirty-six years, Cyanamid has maintained deliveries world-wide ...a record of dependable service which we will do everything humanly possible to perpetuate.

4. PROCESSES

Since crushing and fine-grinding are so large a part of over-all milling costs, elimination of gangue at the coarsest possible size offers a fruitful field for cost cutting.

Cyanamid offers the two most advanced and efficient mechanical processes for this purpose: *Heavy-Media Separation Processes* and *Dutch State Mines Cyclone Separator Processes.*

Both employ unique, exclusive principles to achieve accurate, low-cost separation not obtainable by older methods. These processes have been used on millions of tons of a wide range of minerals as low-cost methods of pre-concentrating ore or tailings prior to froth flotation or other treatment, and to produce directly-marketable concentrates.

*Reg. U. S. Pat. Off.

**AERO is a trade-mark of American Cyanamid Co., applied to Xanthates, promoters, depressants, reagents, etc., for use in the flotation of minerals.

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AMERICAN Cyanamid COMPANY

MINERAL DRESSING DIVISION

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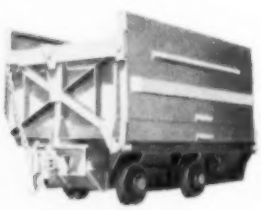
for a safe, smooth ride . . .

THAT'S what this Lake Shore Man Haul car (above) gives 30 men . . . a dry, safe, comfortable, upright ride with plenty of knee and headroom — space on the seats for dinner boxes. Enclosed design gives men full protection. Available insulated against electricity. Car equipped with full universal oscillating trucks, or conventional type.



for heavy loads and rough service . . .

250 cubic feet — the capacity of this tough, all-welded Lake Shore Granby type side dump car . . . designed for the extremely hard loading and operating conditions met in modern, high-production mining.

a standard of reliability . . .

This 38 cubic foot Lake Shore Standard side dump car has given years of trouble-free service in mines all over the country. Time-tested, simple, sturdy design.

Write for more information on Lake Shore "job-proven" Mine Cars and Mine Equipment

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Standard and Special
Mining-Milling Machinery for Every Application.

LOHED MINE CARS

JETO BOTTOM DUMP SKIP

BICYCLE SHEAVES

LAKE SHORE Engineering Company
IRON MOUNTAIN MICHIGAN

Drift of Things

THANKS to ingenious sales promotion letters from a power shovel manufacturer the office has a collection of three cellophane envelopes containing chrysotile asbestos, iron ore, and copper ore, respectively. Now, most men in the mining industry cannot resist collecting ore specimens (and most wives cannot resist throwing them out), but these three samples tell more of a story than usual. The asbestos is from the hills of eastern Quebec, where ever larger mining equipment is needed to handle the tonnages resulting from increased demand and decreasing ore grade. The copper ore is from Arizona, but it could have come from Chile, or Nevada. And the iron ore came from Canada, though it could have come from Minnesota or Venezuela.

What these envelopes of ore do show is the amazing vitality of the industry in finding ore to meet demand and even to anticipate demand decades hence. Most recent evidence of this is the rapidity with which the Bathurst, New Brunswick, find is changing into an established district. In this month's MINING ENGINEERING, p. 1037, there is a report of an AIME fall meeting in Hibbing, Minn., where MBD members had a chance to hear about, then see for themselves, the results of taconite research and the growth and increasing refinement of treatment of intermediate grade iron ore. In October *Journal of Metals* is devoting a special issue to the Colorado Fuel and Iron Corp., a firm that has reversed the usual dictum and is growing eastward. MINING ENGINEERING will have the mining side of the CF&I story in November. To help ME's readers keep up with the "Great Ore Rush," December will be a Famous Mining Enterprise issue on The New Jersey Zinc Co., a company which is opening new properties in Tennessee and Pennsylvania, and exploring actively in Arizona and elsewhere. (R. A. B.)

"BUSINESS people are accustomed to reading and writing reports. The essential difference between writing a report and writing an essay or an article is this: in the report one is charged with conveying information to someone who is, in most cases, already interested, while in an article one has first of all to catch attention, then hold it, and then satisfy it."—From the *Royal Bank of Canada Monthly Letter*, August 1953.

For "business people" read *engineers*, and for "articles" read *technical papers* and you have one of the briefest statements possible about the gap that must be bridged by technical men when they prepare papers for presentation and publication. Subject matter with news type interest—a new plant, or mine, or process—or facts with a strong bread-and-butter value carry many papers. But far too many engineer-authors do not bridge the gap, for they fail to do the first thing when writing an article. They fail to catch attention. Still fewer try to hold it, although painstaking reading will almost always reveal more than enough raw material to satisfy.

Some may argue that technical papers need not meet the readability standards required of articles. Certainly the material does not lend itself to the light, the sensational, or even the easy reading style. The author of technical or scientific material has not an easy job, there is no sex, no emotionalism to fall back on. But a paper that draws the reader to its conclusions is certainly a better effort than that which merely tabulates undigested data. (R. A. B.)

UMPIRES have no friends—not even dogs will ante up the milk of human kindness to the guy in the blue suit. One of the joys attending a Sunday afternoon in Brooklyn is the right to go to the park and call an umpire all the names generations of ball fans have devised. Let the umpire bend his weary body toward home plate to dust the target and some mighty-lunged fan is sure to bellow, "Try a seeing eye dog," or some such home grown witticism. But down in Texas the Plymouth Oilers of the Plymouth Oil Co. have shown mercy. The umpire stands a chance of at least saving a few bones in his aching back for old age. Jesse Thompson, business manager of the Oilers, has installed a push button plate duster that employs compressed air to clean off home plate. The umpire just steps on a valve and a jet set in the middle of the plate does the rest with compressed air. If this should get around, there is liable to be just a little less joy in Brooklyn, pennant or no pennant. (M. A. M.)

HAVING had the distinct pleasure of attending two fall meetings of the Mining Branch within the course of two weeks, the earlier prediction we made about the "bull being taken by the horns," is really coming to pass. As you see in the pages of this issue the Minerals Beneficiation Div. Annual Fall Meeting at Hibbing, Minn., was considerably more than a success. Registration topped the 500 mark, and all participants were well satisfied with the affair. The meeting was well organized and went off with clockwork precision.

Within a matter of four days we were on the scene at the Industrial Minerals Div. Fall Meeting in Nova Scotia. Some 250 members and wives attended the joint meeting of the Canadian Institute of Mining and Metallurgy, Mining Society of Nova Scotia, and the AIME at beautiful Keltic Lodge, Ingonish Beach, which ended with a sail on Halifax Harbor and a lobster broil. This also was a smoothly operated meeting, interesting technically and socially. Hospitality reigned at the gathering.

We have the sincere belief that these meetings preview the pattern to come in fall meetings for the various divisions of the Mining Branch. Intimate meetings of this type permit members to become better acquainted, and strengthen divisional and Institute ties.

Charles M. Cooley

Induction Heating——

For Better Blast Hole Drill Bits

Induction heating, together with automatically controlled tempering and hardening is giving Oliver faster and more accurate bit sharpening, while experiments with bit taper promise to offer increased footage per bit.

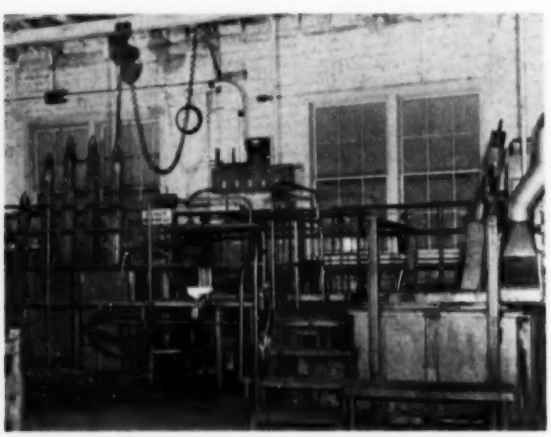
by John H. Hearding, Jr.

LATE in 1947 it was decided to examine the practice followed by the drill sharpening shop in the Hibbing-Chisholm district of the Oliver Iron Mining Div. to see what improvements should be made, in order to obtain more uniformly sharpened drill bits and, if possible, to increase the footage obtained from each individual bit.

At that time the bits were heated to a forging temperature in an oil-fired furnace, forged, allowed to cool, and then reheated in another oil-fired furnace for hardening. The latter furnace required that the drill shop attendant use his eyes to match the color of the bit with the color of the furnace lining to determine when the bit was at the right temperature for quenching. A pyrometer attached to the furnace indicated the temperature of the lining.

Due to the physical limitations of the shop, as well as the human limitations, about sixteen 9-in. bits were sharpened per 8-hr shift by a two-man crew. This low production often necessitated more than one shift in the drill shop, and frequently two crews on at least one of these shifts.

JOHN H. HEARDING, Jr., is General Superintendent, Oliver Iron Mining Div., U. S. Steel Corp., Hibbing, Minn.



View of the drill sharpening shop shows the hardening section with new equipment; left, lead hardening bath; center, quenching tank; and right, the oil tempering bath.

It is well known that drill steel cannot be raised to a forging heat too rapidly without damaging it, if ordinary methods of heating are used. Under these conditions the outer part of the steel expands faster than the inner part and serious cracks will usually develop. A 9-in. drill bit requires between 35 and 45 min to heat safely to the forging temperature. The length of time needed to bring a bit to the proper heat provided a very definite limit to the number of bits that could be heated in one furnace during an 8-hr period.

As previously stated, the hardening temperature of the bit was gauged by the sight of the attendant in the drill shop. Since there is a wide variation in judgment of color between different people and, also, a considerable variation in the same person at different times, there was a great lack of uniformity in the hardness of the bits coming out of the shop. With these things in mind, it was decided to select new equipment for the drill shop which would eliminate, as far as possible, human judgment and increase the number of bits turned out per shift by a single crew.

Various firms supplying heat-treating equipment were consulted and the experiences of other mining companies were evaluated, in an effort to determine what process best suited our needs.

The use of induction heating to bring drill bits to a forging temperature seemed very attractive from the outset, because the time required to heat a 9-in. bit by this method was one third to one fourth the time required when using an oil-fired furnace. This method permits the bit to be heated very rapidly without any damage to the steel.

Manufacturers of induction heating equipment were not at all sure that a proper type of coil could be designed for a blast hole drill bit, because of its varying diameter and contour. One manufacturer agreed to experiment with the design of the coil and test it on drill bits at its factory, in order to determine whether it would work satisfactorily.

At least two means of reheating the bits to a hardening temperature were available and each appeared to offer many advantages over the method

previously used. Some mining companies, as well as manufacturers of bits, were using a salt bath to bring the bits to a hardening heat. But, from the outset, one manufacturer felt that a lead bath offered more advantages than did a salt bath.

After a good deal of consideration, it was finally decided to purchase an induction heating unit to bring the bits to a forging temperature and a lead bath for reheating them to a hardening temperature. By the time this conclusion was reached the Korean emergency had arisen and it was virtually impossible to get many of the items needed for this new process. Because of this situation the new equipment was not put into service until early in 1952.

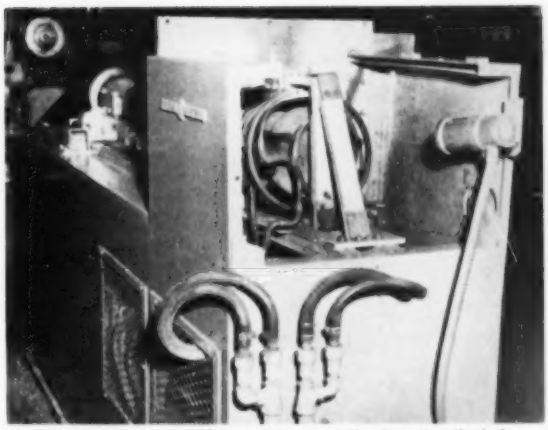
In making the request for this new equipment, it was determined that, if an increase in footage per bit amounting to 25 pct could be obtained, the expenditure would be recovered in less than five years. Selection of the new equipment was made, having two goals in mind: 1—the production of sharp drill bits of uniform hardness; and 2—a 25 pct increase in the footage drilled by each bit.

A brief description of the present sequence of operations performed in sharpening and hardening the bits follows:

Dull bits returned from the mines for sharpening are put into the drill shop through an opening in the wall. They are then laid on a rack built of rails with steam radiators installed below. This rack serves to take the chill off the dull bits during the winter, as they should be at least room temperature before going into the induction heating unit, and also to dry any mud clinging to the bits. Each bit, after it is dry and before it is heated, is thoroughly cleaned on the rack made to hold one bit. This operation was found necessary to prevent dirt accumulating on and under the induction heating coil.

The bit is then transferred to the induction heating unit, where it takes approximately 10 min to bring it to the forging heat. When the correct temperature is reached; i.e., 2200°F, the unit shuts off automatically and the bit is taken to the drill sharpening machine. Most bits are forged to the required size and shape in one heat. Occasionally, if the bit has been badly worn, two or more heats are required to sharpen it.

From the drill sharpening machine, the bit is laid on a cooling rack and allowed to cool naturally to a black heat. The time required for this to take place is approximately 1 hr.

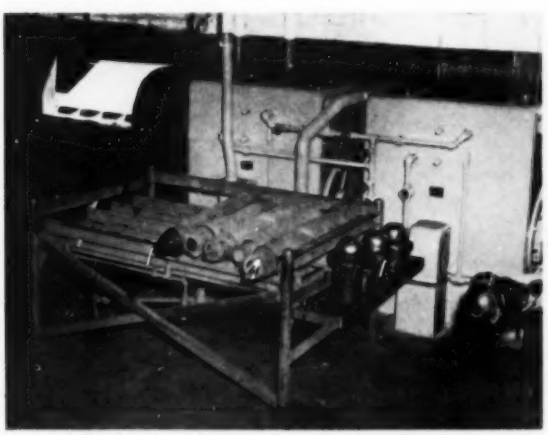


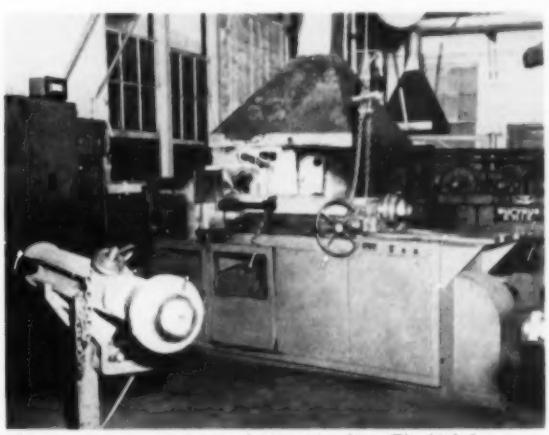
This view shows a drill bit in the specially shaped coil of the induction heating unit. Cooling water connections for the coil are in the foreground.

After reaching a black heat, the drill bit is suspended vertically in the lead hardening bath. The end of each bit is immersed from 2 to 3 in. in the lead bath and is allowed to remain for 30 min. This bath has five stations, each of which will accommodate one bit. Each station is equipped with an automatic timer and signal device, which notifies the drill shop attendant at the end of the 30-min period when the bit has reached the temperature of 1550°F and should be removed.

Next, the bit is suspended vertically in the water quench tank. The end of each bit is immersed to a depth of 12 in. and allowed to remain in the water for 3 min. It was soon found that a rising current of water against the cutting face of each bit was required to eliminate the formation of steam pockets against the face, resulted in soft spots on the bit.

Following this operation the bit is removed and suspended vertically in the oil tempering bath to a depth of 5 in. It is allowed to remain in this bath, which is kept at a temperature of 350°F, for 60 min. An automatic timer and signal light at each of the ten stations in this bath indicates when each bit is ready to be removed.





LEFT: Opening in shop wall for dull bits. The heated rack in the foreground warms and dries the incoming bits. The high frequency motor-generator set is in the background. RIGHT: Rack for cleaning bits is in the foreground; induction heating unit, center. Oil-fired standby furnaces are in the background.

Special Equipment Required for Induction Heating and Automatically Controlled Hardening and Tempering

1— Motor-generator set, with 150-hp, 440-v, 3-phase, 60 cycle induction motor, driving a 100-kw, 440-v, 1-phase, 9600-cycle ac generator.

2— High frequency induction heating unit for bringing blast hole drill bits to forging temperature rapidly.

3— Lead hardening bath, 75 kw, with automatic temperature control and five stations for bits, each with individual timer.

4— Oil tempering bath, 32 kw, with automatic temperature control, and ten stations for bits, each with individual timer.

After the bit has been in the oil tempering bath for the required time, it is then removed and is ready to be returned to the mine for use.

As is usual with any new process, certain difficulties developed. After some time it was found necessary to change the shape of the induction heating coil, in order to heat more of the shank of the bit and still not burn the tip.

The greatest amount of difficulty experienced has been with the pot for the lead hardening bath. This was due chiefly to a very severe corrosion which attacked the nickel alloy pot at the surface of the molten lead. In a period of about nine months, corrosion at the lead surface penetrated a depth of $\frac{5}{8}$ in. into the side of the pot. To date no satisfactory solution has been found for the corrosion of the original pot. A second pot is now in service, but it is too early to determine whether or not the corrosion problem has been overcome.

General shop view shows rock for cleaning and drying bits, induction heating unit, with standby equipment in the background.

The surface of the molten lead must be covered to prevent the forming of excessive dross. Originally fine coke was used, as well as spent charcoal. It is thought that the severe corrosion of the original pot may have started because of an excess of sulphur in the coke, but there is no way to prove this. At the present time graphite and spent carburizing compound are being used to cover the surface of the molten lead with good success.

The new process requires a three-man crew, who sharpen 25 or more 9-in. bits in an 8-hr shift.

Our figures indicate substantial annual savings for the induction heating and automatically controlled hardening and tempering process, as compared to the oil-fired furnaces formerly used. These savings were based on 274,748 ft of rock drilling in the Hibbing-Chisholm district with 9-in. blast hole drill bits during a 10-month period in 1952.

The savings indicated have not been quite as large as expected, because of a change made in the shape of the drill bit. The result of this change is explained later.

There was a small savings in the field, due to the use of fewer bits, with a consequent reduction in labor required to change them. The fact that fewer bits were used also reduced the number of trips required by trucks and other equipment to deliver and collect the bits at the mines.

The rock drilled in the Hibbing-Chisholm district is divided into four types:

Paint rock, a soft red phase of the gray slate, which drills easily.

Slate, a fairly soft gray rock that is slightly harder to drill than paint rock.

Stratified taconite, a harder rock with some thin soft seams, frequently badly broken. It is considerably more difficult to drill than slate.

Blocky or solid taconite, a rock that is very hard and difficult to drill.

In an attempt to obtain an increased footage per bit in the blocky taconite, the taper on the reaming surface of the bit was made 2° as compared to 15° formerly used. This resulted in practically doubling the footage per bit in the blocky taconite, but caused a substantial decrease in the other three types of rock. The loss in footage was not caused by the drill bit becoming dull more quickly or losing gauge, but was due to a tendency to stick in the soft material when a smaller taper was used. Once a tendency to stick occurred, the driller would remove the drill bit, even if it were not dull, rather than risk having it become stuck in the hole.

Since the drilling of blocky taconite constitutes only about 15 pct of the total rock drilled, the loss in footage per bit in the other three materials greatly exceeded the gain in the blocky taconite. At present the amount of taper on the reaming surface is being increased by small amounts, in order to arrive at a compromise between greatly increased footage per bit in the blocky taconite and a loss in footage in the other three materials. There is every reason to believe that a taper can be found which will give fairly satisfactory results in all four types of rock. When this has been determined, there should be no difficulty in obtaining an increase of 25 pct in the footage drilled per bit over the entire amount of rock drilling.

In addition to the gains listed above, there have been other advantages in the change to induction heating. The shop is much cleaner, less noisy, and a safer place in which to work. The intense heat from the oil-fired furnaces, which was a problem in the summertime, has been eliminated.

After slightly over a year's operation with the new type of equipment, the feeling is that the equipment was well chosen.

Thanks are due Mr. C. R. Burton and Mr. J. R. Schoenig of the Maintenance Department for their assistance in the preparation of this article.

Summit Develops Pennsylvania Nonmetal Mine

WHEN geologists began probing the grayish rock mass, natives of Adams County, Pa., called *soapstone*, they were paid scant attention. However, the soapstone, at first misidentified as pyrophyllite, was in reality sericite schist, a hydrous aluminum silicate whose chemical analysis resembles pyrophyllite, except that it has a slightly higher iron and potash content. In addition to its grayish color, sericite schist is foliated and has a greasy feeling much like that of talc.

In 1947 W. T. Weldie, former clay producer from Dillsburg, Pa., prospected the area around Adams County for additional clay reserves. He chanced upon what was first thought to be a prophyllite deposit, 3½ miles from York Springs. Diamond drilling disclosed quartz and other impurities which rendered the deposit unfit for commercial production.

Discovery of other deposits in Adams County led to the organization of Summit Mining Corp. in June 1950. Since that time Summit has been busy prospecting, testing, and developing, with over 40 company leases spread throughout Adams and York Counties. At the moment, only one lease is being worked. Called the Heller property, it is similar to other deposits in that it forms the crest of hills. Terrain features and natural drainage make open pit mining expedient. Sericite schists are found in belts along many of the contacts of the metabasalt and aporhyolite in places where movement and shearing have evidently taken place. The Heller deposit outcrops for 600 ft and is 75 ft wide at one point. The company has no definite knowledge of the depth of the deposit, but additional knowledge may be garnered from prospecting now in progress.

A bulldozer was employed for first stripping operations. However, as mining progressed back into the hill, shovels and trucks were used. Jackhammers have carried the drilling load, but a wagon drill is expected to be operating shortly. Delayed electric caps and 60 pct gelatin dynamite are used for blasting. Initial experiments with 40 pct gelatin dynamite returned insufficient shattering effect because of the laminated structure of the rock. Future experimental work entails development of correct drill hole pattern and proper explosive for the most economic operation.

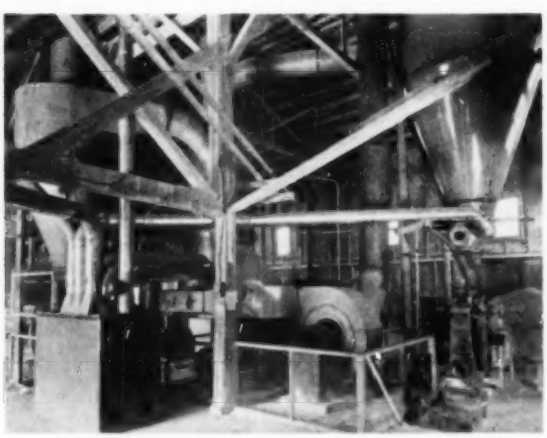
Information for this article was supplied by GEORGE AHL, Jr., Assistant to the President of Summit Mining Corp., and C. M. DOLLIVER, Jr., Production Manager of Summit.

Bulldozers and shovels are used to mine Summit's Heller property. A wagon drill is planned for the near future, but at the outset of operations jackhammers bore the brunt of the drilling chores.

Enough material can be turned out of the primary crushing unit in a week to last the mill a month. Thus, mining and hauling is on a contract basis. A ½-yd shovel mines the ore and loads it into 10-ton International and White dump trucks. The arrangement has proven satisfactory because mine operations can be carried out in good weather. A reserve stockpile at the mill is maintained at 1000 tons for use during adverse winter conditions.

The processing plant is at Dillsburg, 22 miles from the mine. It consists of primary and secondary crushing units and pulverizing units. Truck loads are discharged into a pan feeder which routes material into a 12x26-in. American Road Machinery jaw crusher. Discharge from the jaw crusher is fed to Scottdale 18-in. double tooth rolls. The −1 in. material goes to the stockpile via chain conveyor. Primary and secondary crushers and the stockpile are housed in the main mill building. A Case front-end loader carries the material to a bucket elevator which delivers the material to the mill feed bin. A Standard Williams roller mill with kiln attachment does the pulverizing. With the kiln, the final product has less than 0.3 pct moisture, and because additional moisture is not picked up, drying is only a slight problem.

The finished product is bagged in No. 50 pasted bags by a Bates bagger. The mill is turning out a 93 to 95 pct −325 mesh product for current users. Some of the major insecticide companies are using the material in their blends. Results have been excellent in plane dusting and in other field procedures because of superior drift control properties, flat particle shape, and neutral pH. It is also nonhygroscopic. Certain static properties inherent in the sericite have made it satisfactory in the rubber dusting field. It eliminates blotting when mixed in various proportions with certain talcs. Research is being conducted in the use of the material for fillers and ceramics.



Ore from the Heller mine goes to the mill at Dillsburg, some 22 miles away. Enough material can be turned out of the primary crushing unit in a week to last this custom grinding plant a month. The processed sericite schist, trade-marked Ser-X, finds use as insecticide dilutent, for rubber dusting, and as a filler material.

Carryall Scraper Plays Major Role In Open Pit Mining at Inspiration

by Herbert H. Mellus

TODAY the tractor-drawn carryall scraper has a very important role in the Open Pit div. of Inspiration Consolidated Copper Co. Carryalls have already moved over 3 million tons of material at costs comparable to or lower than for conventional truck and shovel operation, and a major program of scraper stripping and ore production schedules has developed from early isolated scraper applications.

Application of carryall scrapers to open pit production at Inspiration began in August 1948 when a LeTourneau model FP tractor-drawn carryall scraper was purchased. A bench approach road 30 ft wide and 1400 ft long with cuts up to 45 ft was planned, and it was decided that a scraper could be substituted for the Dart truck and diesel shovel operation even on hauls over 400 ft. It was also thought that a scraper could be employed to good advantage in inital development of benches, where material from several wagon drill blasts had to be moved to open up a full face for normal shovel and churn drill operation.

First major scraper stripping project was initiated in November 1948 when it was decided to strip the upper benches of the as then planned Colorado pit by tractor-drawn scraper. High cost of haulage road construction combined with the small tonnages involved on the upper benches made it appear inadvisable to consider a conventional shovel operation. Waste disposal areas within a level haul of 400 to 800 ft were available for scaper operation. Using the FP scraper drawn by a Caterpillar D-8 tractor (with one additional D-8 for pushing, necessary ripping of hard areas, and maintenance at the waste disposal area) an average of 960 tons removal per shift was maintained. With ground previously broken by a churn drill blast and a comparatively level haul to waste dumps within 400 to 800 ft of the loading site, an average of 40 trips per shift with a load of 24 tons could be expected from the LeTourneau Model FP carryall scraper. About 325,000 tons of waste were removed in this manner before the project was temporarily shelved because of the moving back of the pit slopes to incorporate the larger Thornton pit.

In December 1950 a stripping project in the Live Oak pit was started to remove the overburden above several of the 800-level blocks of the underground caving operation. Initial stripping was scheduled for scraper operation and a Caterpillar "80" with 20-cu yd heaped capacity was rented to supplement the 18-cu yd capacity FP scraper. So satisfactory was this project that in June 1951 Inspiration purchased a Caterpillar "80" scraper and increased its heaped capacity to 23 cu yd by installing 12-in. sideboards. During the next year the FP and "80" scrapers were used advantageously on small stripping operations and road building in Live Oak and Thornton pits.

In February 1952, upon completion of the Block No. 1 ore pass in the Thornton pit, dozers were used initially to keep the block supplied with ore from already broken reserves lying adjacent to the block perimeter. It was apparent early in this operation that it was necessary to keep the hole filled to prevent sloughing from the walls allowing large blocky

For the full story of Inspiration's use of a caved block as an ore pass from the open pit see the article by H. Carroll Weed, MINING ENGINEERING, September 1953, p. 886.

HERBERT H. MELLUS is Pit Engineer, Inspiration Consolidated Copper Co., Inspiration, Ariz.

One type unit used by Inspiration is the Caterpillar "80" scraper with two D-8 tractors, shown here stripping a lead-zinc open pit in Missouri.

The LeTourneau FP scraper, shown on a large earth moving job in this view, was the first type scraper used by Inspiration.

Loaded carryall scraper is approaching partly drawn Block No. 1 ore pass. Dozers were used to keep ore pass filled until distance to ore exceeded 200 ft, then scrapers were used until July 1952. Transfer was kept full to prevent sloughing from the walls.

material to interfere with the underground draw. Material supplied by the dozers would run freely in the underground drawpoints unless dampened by surface moisture. The tractor-drawn carryall scrapers were employed as soon as distances between available ore and the block opening exceeded 200 ft. Until July 1952 the LeTourneau FP and Caterpillar "80" scrapers were used for this project; then two Caterpillar "90" scrapers with 31-yd capacity were obtained.

In an eight-month period 654,000 tons of ore were delivered to Block No. 1 by scrapers; the FP, "80", and "90" scrapers averaging respectively 1580, 1730, and 1830 tons per shift over round trip hauls between 600 and 1600 ft. "Round trip haul" designates the distance traveled from the start of the loading operation through the complete cycle to the dump and back to the loading point again. In scraper operation the return route may vary from the loaded haul route and influence trip time considerably.

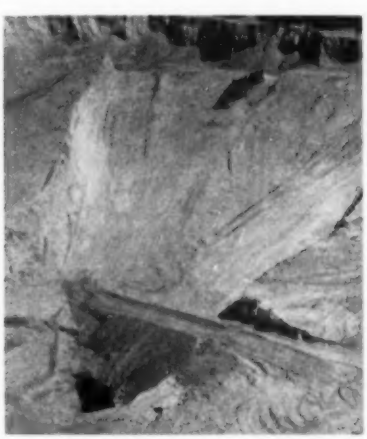
During the Block No. 1 operation the "90" scraper averaged 10 tons more per trip than the "80" model, but its advantage was partly offset on shorter hauls by the greater loading time required. Where the "80" scraper would load readily in good ground with the assistance of a push cat, the "90" scraper loaded more slowly due to its heavier weight and construction. As a result the "80" scraper averaged 10 pct more trips per shift than the "90" on this operation.

During October and November 1952 the scrapers were used on stripping where the round trip haul averaged 5000 ft over comparatively level ground. Here the "80" and "90" scrapers hauled 640 and 800 tons per shift respectively, showing that the "90" could be used to a greater economic advantage as the haul distance increased.

Considering the results from these previous operations it was decided to strip the upper three benches of the Thornton pit by scraper. This would involve removing of 940,000 tons of waste and placing it in small canyons and along steep slopes outside the pit perimeter. Constructing a haulage road to this area for truck and shovel operation would have entailed too high a development cost for the amount of material involved. This project started in January and is now in progress. Primarily the "90" scrapers have been used, and 442,000 tons had been removed by May 1. At that time it was decided to continue stripping by scrapers for an additional 50 ft below the originally scheduled operation; upon completion it is expected that 1.8 million tons will have been removed by scraper before normal shovel and truck operation takes over. Round trip haul has varied from 600 to 2400 ft and the "90" scraper averaged 1350 tons per shift at an average load of 35 tons per trip for the first four months of operation.

Loading conditions have not been ideal as ribs of hard ground have been encountered that resulted in blocky material that interferes with loading. The blocky material also interferes with closing the apron tight enough to prevent spillage of fines. With good conditions 40 tons per trip could be hauled by the "90" scrapers instead of the 35 tons averaged so far on this job.

Evidence that carryall scraper application in open pit mining has proven to be an integral part of the operation at Inspiration is the fact that over 10 million tons of ore and waste are scheduled for removal by tractor-drawn scrapers during the next five-year period.



LEFT: Block No. 1 ore pass is partly drawn. Note scraper near top of slope in this view. RIGHT: General view of pit shows one scraper delivering to the transfer (dotted outline) which is full. A second scraper with pusher tractor is loading in foreground.

Geochemical Prospecting Finds Widespread Application in British Columbia

by Harry V. Warren and Robert E. Delavault

IN the worldwide search for buried orebodies and for entirely new areas of mineralization, geochemistry is taking on an increasingly important role. Indeed, geochemistry itself is splitting into various specialties—hydrogeochemistry, pedogeochemistry, and biogeochemistry. This article primarily relates to developments in these fields in British Columbia, but many of the remarks are applicable to the whole continent.

There are several factors in the background of geochemistry in British Columbia. First, virtually every gold and base metal mine producing in 1953 was discovered more than 50 years ago. Technological improvements and increased prices for base metals have resuscitated former camps, maintained and expanded old mines, and brought about re-examination of countless previously known properties. In short, mine finding has been relatively unsuccessful for several decades.

Second, most of the Pacific Northwest is, physiographically speaking, young to mature in character and has only recently, geologically speaking, emerged from the ice age. Thus the area has rugged topography, comparatively primitive transportation facilities, and relatively few areas where the soils have mature profiles.

Third, in British Columbia at least, geochemical researches have been largely empirical and confined to investigations which offered reasonable expectations of producing results which might be exploited.

Hydrogeochemistry

Anybody who prospects in the mountainous regions of British Columbia is impressed by the number of streams and rivulets. In the past prospectors panned these creeks religiously in their search for gold and later for mercury, tungsten, and tin minerals. With a sufficiently sensitive technique for detecting traces of metal carried in solution, these same creeks might be used to detect various types of orebodies, providing the orebodies were undergoing oxidation or solution. Unfortunately in most of the mountainous areas, snow, ice, and rain dilute the natural water to such an extent that even Huff's sensitive dithizone test for heavy metals in water proved impractical for general prospecting. However, the idea was too intriguing to drop and by means of an emulsion technique developed by Dr. Delavault it became possible to detect as little as one part in one thousand million of copper or zinc in natural water, using only a light field kit.

This test was first put to practical use in the Tweedsmuir Park area, about 400 miles north of Vancouver. After about a month of negative results two positive creeks were discovered. One, named Two Gamma Creek, could not be considered a discovery because it led back to a well-known prospect. When followed a mile upstream from a lake the zinc content of the creek water, by there shorn of most negative branches, commenced to increase rapidly, and by the time the old showing was reached the creek carried more than 100 gammas of zinc, i.e. more than 100 parts per thousand million. The second creek, named Four Gamma, led to a small stringer of oxidizing chalcopyrite crossing the creek about a mile above its mouth. The creek was negative above the stringer. These two creeks were tested on many occasions, and from them it was learned that uniform results should not be expected, either from day to day, or even from hour to hour. The relationships between the zone of oxidation and rise and fall of the water table seemed to be of great importance to the amount of metal in a stream at any given time. Hourly changes seemed related to rapid changes in run-off.

As a result of tree and plant samplings two more streams were revisited the following field season. One, now named Anomaly Creek, revealed a shear zone containing copper. Direct observations showed the mineralization to be too restricted in area to be commercial. On a second creek, "purple water"—i.e. carrying over 5 gammas of zinc—was followed 3 miles back and some 2000 ft above the elevation of Eutsuk Lake. Between 15 and 20 tributaries were eliminated on the way and the zinc content rose to 40 gammas. Once again mineralization proved noncommercial. Nevertheless Purple Creek proved a point, for it would have been negative to all the tests available to a prospector prior to development of the emulsion technique. It is interesting to note that no zinc bearing float was recognized farther than a quarter of a mile from the mineralization.

During the past 2 years much water has tested positive, but most of it has merely led to rediscovery of old and long abandoned workings. This is not surprising because most of the testing has taken place around old mining camps to establish water testing as a practical prospecting method.

Toby Creek in the Windemere District and Kokanee Creek near Nelson on the west arm of Kootenay Lake provide textbook illustrations of water testing as a prospecting tool. Kokanee Creek has two mineralized bands of pyrrhotite with minor sphalerite crossing it, one 2 miles above Kootenay Lake and the other 5 miles farther up. These two bands of mineralization, well below commercial grade at present, are sufficient to salt Kokanee Creek during spring and autumn when the creek is high, but not during the summer when it is low. This is believed caused by the water table in this mountainous country falling below the zone of oxidation in the summer, thereby cutting off any effective contact between the bulk of the mineralization and the creek. Toby Creek has a small tributary which originates near the Paradis mine, at an elevation of about 8000 ft. This tiny creek, which probably flows only 4 miles before joining Toby Creek, carried more than 10 gammas of zinc on the day it was tested. This amount of zinc was ample

HARRY V. WARREN is Professor, Dept. of Geology & Geography, University of British Columbia, Vancouver, B. C. ROBERT E. DELAVAULT is a Research Associate at this University.

for recognition of this metal in the tributary, but not sufficient, at that time, to salt Toby Creek in recognizable amounts.

In short, the emulsion technique of water testing has extended hydrogeochemistry as a prospecting tool: it enables natural water to be prospected in much the same manner as stream gravels were once tested for gold colors. Finding colors in a stream was a good sign but did not necessarily mean that ore was nearby. Similarly, finding copper or zinc in water does not mean that there is ore in the vicinity. In either case further prospecting is merited.

Pedogeochemistry

Soil sampling should prove to be an increasingly useful tool in Canada, even under the generally recognizable handicaps of irregular rock surfaces and immature soil profiles. (Editors Note: *Testing for Copper and Zinc in Canada Glacial Soils*, by C. T. Bischoff, will appear in the December MINING ENGINEERING.) Sufficient work has been done by the authors to say that in the vicinity of either copper or zinc mineralization anomalous amounts of the appropriate metal may be expected in the soil.

Biogeochemistry

The authors have already published many of the results of ore seeking using biogeochemistry. Results of some projects have not yet been released for publication. Unfortunately no big operating mine can yet be pointed out that was found entirely by biogeochemical means. Actually only necessity would cause the authors to recommend drilling on biogeochemical results alone: wherever possible supporting geological or geophysical evidence should be integrated with biogeochemical results. However, as a result of the past 8 years work the abandonment of some ground where the biogeochemical evidence was unfavorable can be pointed out. Biogeochemistry assisted in varying degree in the finding of mineralization in some drift covered areas. It might well be asked, "How does one know that the abandonment was right?" In one instance a bulldozer was brought in and stripping carried out only to find that the projected mineralization did not continue, and did end as had been indicated by the biogeochemical survey.

In another instance in an effort to carry out a systematic drilling campaign a negative anomaly was drilled and the hole was abandoned after going through 150 ft of what turned out to be an unexpectedly thick glacial moraine. Experience suggests that where the overburden consists of more than 30 ft of muskeg or glacial material in which clay is predominant, biogeochemical results are going to be inconclusive, except under most favorable circumstances.

Some species of trees have the ability to collect relatively large amounts of some element present in much smaller amounts in soil, and such trees may be used in that particular locality in preference to the soil. Trees possessing this ability include Silver Birch *(Betula papyrifera)* for zinc, Hemlock *(Tsuga heterophylla)* for manganese, and Balsam *(Abies lasiocarpa)* for molybdenum.

Even digging will yield little information here, but soil or tree sampling may delineate anomalies.

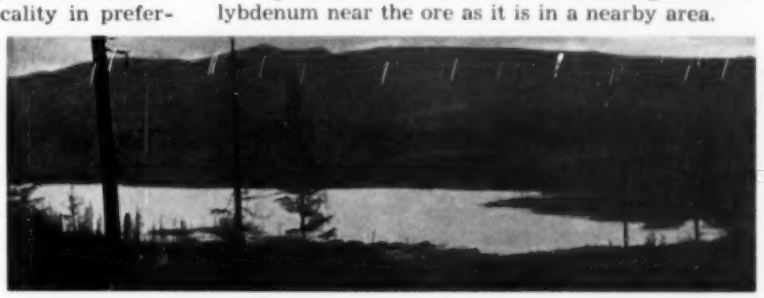
Where the drift cover is less than 30 ft it is seldom that a suitable collection of vegetation fails to reveal the presence of underlying mineralization. Whether soil or tree sampling will be the better way of seeking to outline or discover an anomaly would depend on circumstances, with ease, speed, and accuracy of sampling of prime importance.

However, biogeochemistry must not yet be expected to indicate the grade of underlying mineralization, at least until a considerable amount of work has been done in a particular locality. One of the most promising anomalies yet encountered was obtained with Cedar *(Thuja plicata)* trees, and it led to the discovery of a large amount of zinc, but the grade was not commercial, being less than 0.5 pct.

In 1944 Warren and Thompson started using the term *pathfinder* for minerals or elements found in close association with the element being sought, and which can be more readily found or have more obviously anomalous features than the element which is the main object of search.

The use of molybdenum as a pathfinder for copper may prove to be important. The normal copper content of dry plant material runs from 4 to 10 ppm and for anomalous material, possibly 12 to 24 ppm, or only two to three times normal.

In contrast, molybdenum normally runs only 0.1 to 0.2 ppm in dry plant material. In the vicinity of a very modest showing of molybdenum, Balsam *(Abies lasiocarpa)* was found carrying 150 ppm. Around a disseminated copper deposit, San Manuel, the molybdenum ran 10 to 20 times normal. Molybdenum in small amounts frequently accompanies the porphyry coppers, and in at least one British Columbia copper showing the vegetation for the most part is from five to ten times as high in molybdenum near the ore as it is in a nearby area.



In desolate areas like this one outcrops are rare, and digging at random is hardly profitable.

What Automatic Controls Can The Mill Operator Use?

by James E. Lawver

A SURVEY of Minerals Beneficiation Div. membership indicated genuine interest in automatic process control, but revealed that the average mill is operating with a minimum of self-regulating devices. Apparently, the mill operators are content to continue without controls until the practicality of self-regulating devices is more generally accepted, or, in the opinion of the writer, until automatic controls become a necessity to meet economic demands of product quality and cost.

Both to stimulate interest in control, and to solicit publication of technical know-how by members and equipment manufacturers, this paper presents a cross section of control problems in two general categories: (a) Control problems that can usually be solved by reference to the literature, i.e. controls that have proven successful in the mineral or chemical industries; (b) Problems that are troublesome and frequently unsolved due to unavoidable adverse conditions in the mill circuit to be controlled.

Successful Controls

Liquid Level Control The usual difficulty with this control is that the mill designers select variable head, area meter, or metering pumps several times the peak flow because reagent consumption is not exactly known. The resulting installation is almost useless untill the proper size metering element is installed.

pH Control Controllers for pH are usually successful provided the following precautions are taken:

1. The amplifier should be placed as close as possible to the electrodes in order to minimize the possibility of stray voltages in the input circuit. Good grounding and shielding is essential. All electrical connections should be soldered.

2. If the pulp being controlled is abrasive or contains +65 mesh particles, a miniature thickener or filter should be installed ahead of the measuring circuit.

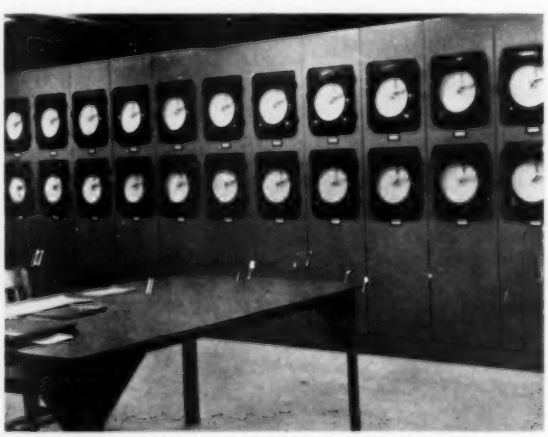
3. Electrodes should not be used that will react with a collector because excessive cleaning will be required.

Combustion Controls on Rotary Kilns The measuring and controlling elements of combustion regulators on rotary kilns and dryers are usually very straightforward and simple to install. However, many installations are unsatisfactory due to the large capacity or transfer lag. In such cases a working knowledge of rate response relays is essential.

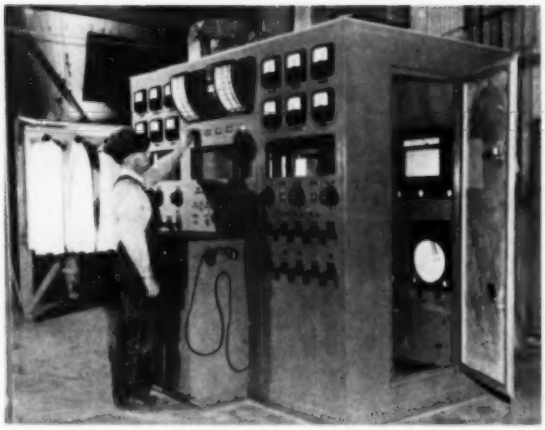
Automatic Flue Gas Analysis Continuous CO_2 analyzers and recorders of continuous Orsat and thermal conductivity types are in general use. Gas density and infra-red absorption units have also been successfully installed.

JAMES E. LAWVER was formerly Research Engineer, International Minerals & Chemical Corp., Mulberry, Fla., and is now at the Graduate School, Colorado School of Mines, Golden, Colo.

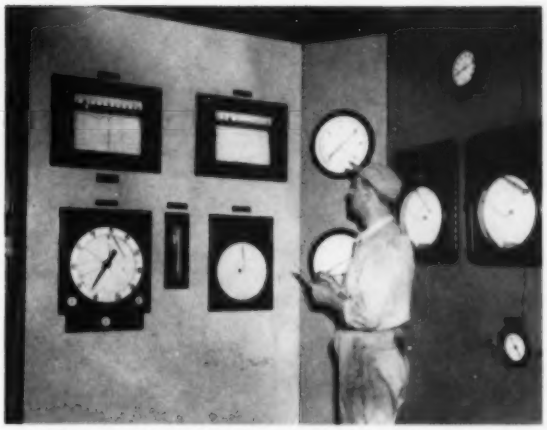
Master control panel in the Noralyn mill provides the nerve center for beneficiation operations at International Minerals & Chemical Corp.'s Bartow, Fla., phosphate mine.



This impressive panel of recording and controlling reagent meters in the Noralyn mill control center is tied to the flow meter assembly in picture on opposite page.



In industries with one major unit operation, centralized control is very feasible, as indicated by this dust-proof, centralized instrument panel for complete control of a rotary kiln installation.



A Dorr reactor is controlled from this Honeywell panel. The instruments in front of operator record reactor temperature, record SO_2 pct in exit gas, control temperature, and record slurry density.

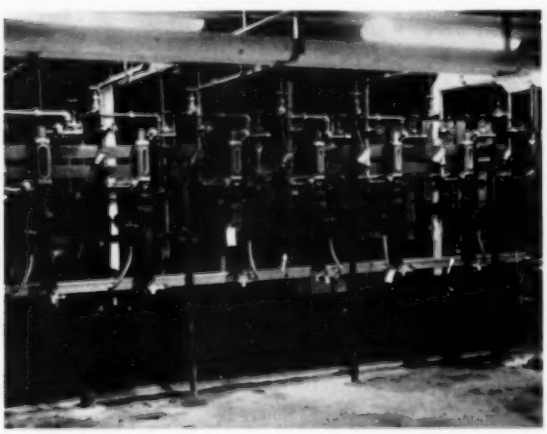
Tonnage Control—Moist and Dry The success of a tonnage control frequently depends on the mechanics of the feeder. If the mechanism is such that an operator can easily control the feed rate manually, automatic control can be successfully installed. Reasonably dry material can be controlled by a number of methods such as the Hardinge Constant Weight Feeder, or by having a vibrating feeder controlled by a load cell on the discharge belt. Wet materials that can be fed by table feeders can be similarly controlled by having an air cylinder operate the feed gate. Either floating or proportional-reset control can be used, and tonnage can be interlocked with control of other variables in the flowsheet.

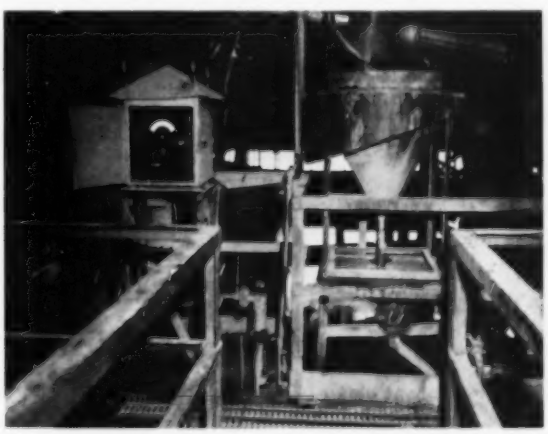
Troublesome Controls

Density control of mixes at high percent solids. In general, the density of a solid-liquid mixture with agitation cannot be successfully measured by the use of bubble tubes or diaphragm units due to random pressures at the point of measurement. If the mixer is a constant level type, the density can be measured (and hence controlled) by weighing the mixer and its contents by means of a pneumatic load cell, or with strain gages.

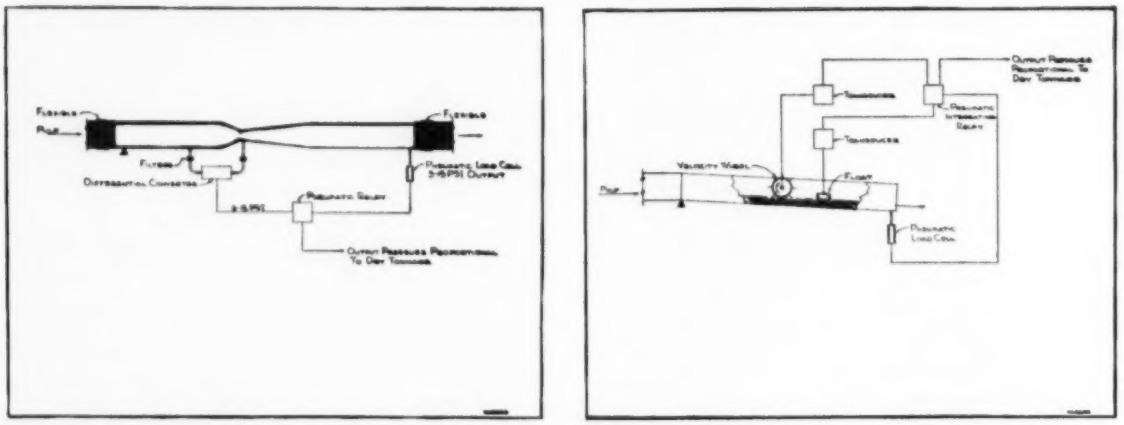
Since mixer power varies directly with density in many applications, it is often possible to control the percent solids by the installation of a thermal watt converter and a standard potentiometer controller. This has the serious disadvantage that any increase in friction after standardization will cause high readings and wear on the mixer blades will cause low readings. Whether or not such a control is applicable can be easily checked by installing a recording wattmeter (not ammeter) and plotting percent solids versus power. This type of measurement cannot be used if the major variation in power is due to something other than the measured variable. For example, the feed to a ball or rod mill could not be controlled by measuring the power load on the mill because variation in power due to the ball or rod load is considerably greater than variations due to the feed rate.

Density Control of Thickener Underflow. The bubble tube method is usually satisfactory if the measurement point does not have in excess of 50 pct solids. In this arrangement the differential pressure tubes are mounted in the discharge pipe, and operate an air cylinder type pinch valve through a pneumatic controller. But, at high solids, random pressures and possibility of thickener discharge choking make the method unsatisfactory.





IN THE PLANT: Flow meter assembly and reagent control at the Noralyn mill is connected to instrument panel shown on opposite page. RIGHT: Automatic pH control is provided by this Brown-Beckman pH measuring system which has dewatering cone ahead of pH cell. The Beckman amplifier on the left transmits the signal to a remote ElectroniK recorder.



The author suggests control schemes for wet milling circuits might utilize either differential pressure readings on a throttle tube, left, or else a combination of velocity wheel and float in a launder, right. These would be interlocked through a pneumatic relay and made to yield output pressures proportional to dry tonnage.

Density Controls in Heavy-Media Circuits Interest in this field is so great (see table), and the problems so varied that a full separate discussion should be devoted to this subject, and no comment is made here.

A survey of Minerals Beneficiation Div. members to determine program desired of the Unit Process Committee showed this distribution of interest—

Specific Request	Pct
Density Control in Heavy-Media Circuits	28
Wet Ball & Rod Mill Grinding	16
pH Control	15
Liquid Reagent Feeding	7
Automatic Product Analysis	6
Mesh Size of Classifier Overflow	4
Density Control of Mixers	4
Density Control in Thickener Underflow	4
Tonnage Control	4
Wet Launder Weightmeter	4
Automatic Flue Gas Analysis	2
Installation & Maintenance Costs	2
General Control Papers	2
Burner Controls on Kilns & Dryers	2
	100

Wet Ball and Rod Mill Grinding Controls It is of interest to note that the survey produced no requests concerning automatic control of dry grinding mill circuits. Wet grinding control circuits, on the other hand, leave much to be desired. Although there are several useful controls, such as the Hardinge Electric Ear, complete automatic circuit control of the average grinding plant is at present impossible due to the lack of two reliable measuring devices:

1. An instrument to continuously determine mesh size of a classifier overflow which could be interlocked with the feed rate to the grinding circuit.
2. A rugged measuring device which will measure the dry feed rate in a mill pulp flow.

There have been several good approaches to the latter instrument, such as the Hagan Ring Balance, and other devices by the Fischer & Porter Co. However, these have not shown successful application in wet grinding circuits.

In the author's opinion, control schemes utilizing either differential pressure readings on a throttle tube, or a combination of velocity wheel and float in a launder, interlocked through a pneumatic relay, could be made to yield output pressure proportional to dry tonnage. It is to be hoped that some of the leading instrument companies will pursue this subject to the point that generally satisfactory measurement and control can be worked out.

Automatic Product Analysis At present this is largely limited to highly radioactive minerals or minerals that fluoresce. Detection of trace impurities by continuous X-ray diffraction analyzers and continuous mass spectrometers is usually completely out of the question for mill control both from a cost and product purity standpoint. It is often possible to develop special automatic analyzers for the particular product being processed, but this invariably requires research and development because the great variety of products which must be analyzed prohibit expenditures on the part of control manufacturers.

In closing, it should be emphasized that before any automatic control can be successfully installed foresight and planning is essential After installation, the control can only be satisfactory if adequate maintenance cost is provided. A good rule-of-thumb for maintenance cost is one man hour per week per $1000 capital investment—in other words, a $40,000 control installation would require one maintenance man working a full 40-hr week.

The author wishes to express appreciation to J. G. Ivy, R. H. Berg, D. Anderson, and W. Barbarowicz for their comments and suggestions, and Mrs. Parkermege for her work on the survey.

References

The author suggests the following references for a thorough basic understanding of control theory:

General Control Theory

A Simplified Concept of Automatic Controller Actions—The Bristol Co., Waterbury, Conn.

Air Control Theory—The Bristol Co., Waterbury, Conn.

Chemical Engineering Guide to Process Instrument Elements—McGraw-Hill Publishing Co.

Industrial Instruments for Measurement & Control—Thomas J. Rhodes.

Principles of Industrial Process Control by D. P. Eckman—Scientific American, September 1952.

pH Control

Principles and Significance of pH Measurements—Chemical & Metallurgical Engineering, August 1940.

Rate Response

Rate Response Relays in Pneumatic Control Circuits—The Moore Products Co.

Block Caving at Bunker Hill Mine

by C. E. Schwab

A lead-zinc orebody, in fairly strong quartzite and with a dip of 35° to 60°, is block-caved by use of scrams in a stair-step pattern up the ore footwall. Scram linings to handle coarse muck and permit the use of folding scrapers are developed by the use of end-grain wooden blocks to reduce maintenance and keep operating cost to a minimum.

THE Bunker Hill mine, since its discovery in 1885, has steadily produced a high grade of lead-silver-zinc ore. By the end of 1952 over 21,000,000 tons of this high-grade ore had been produced by square-set mining, and reserves in the mine continue to be very satisfactory both as to quantity and grade. For many years prior to 1941, mine production and mill capacity had been 1200 tons of feed per day. Closely adjacent to the mill, and stored behind dikes, coarse jig tailings had been impounded during the time preceding the advent of fine grinding and selective flotation. When manpower became short in 1941 and sink-and-float preconcentration was proved successful, mill capacity was increased to 1800 tons per day to treat these jig tailings economically.

By 1946, because the supply of jig tailings was limited, underground exploration was started to discover and prove ore reserves of low-grade material which could be mined by an appropriate bulk mining method. During the years of square-set mining many possible areas of low-grade mineralization had been observed. One chosen for the first exploration work was sufficiently remote from active mining areas so that subsidence, if an orebody were proved, would cause no problem. Also, old adits and workings were still open and in good enough condition so that exploration in the mineralized zone could be started with a minimum of preparatory work.

In 1948 an orebody was proved of sufficient tonnage, of a grade about 2 pct Zn, 0.5 oz Ag, and 1.0 pct Pb. It was decided to use block-caving, the only appropriate mining method by which this grade of ore could be economically recovered. Exploration for additional reserves in other areas of the mine is continuing, but ultimate results are not known at this time. With more sink-and-float capacity, larger ball mills, and more flotation machines, mill capacity was increased to 3000 tons per day, permitting the mining of ore in the square-set area at a maximum rate not usually achieved, because of the scarcity of labor. Increased mill capacity also permits block caving and the mining of jig tailings at variable rates to keep mill feed up to 3000 tons per day.

C. E. SCHWAB is Manager of Industrial Relations and Supervisor of Block Caving, Bunker Hill and Sullivan Mining and Concentrating Co., Kellogg, Idaho.

Discussion on this paper, TP 3621A, may be sent (2 copies) to AIME before Dec. 31, 1953. Manuscript, Feb. 27, 1953.

Fortunately the three types of feed are amenable to the same mill circuit and reagents for recovery of Pb and Zn. For example, during the first 10 months of 1952 square sets produced 827 tons per day, block-caving 1421 tons per day, and jig tailings 643 tons per day, an average daily production of 2891 tons for all three products.

Exploration had proved the existence of an orebody 1000 ft long and 165 ft wide in horizontal section, see Fig. 1. Company engineers were concerned only with the vertical extension, about 300 ft, from an old level to the surface. Much of this almost outcropped, Fig. 2. The ore lies in the hanging wall of a major fault of the Bunker Hill mine, standing at 65° in one end of the zone and separated from the fault by a wedge of waste, see Fig. 3. This wedge pinches out near the center of the zone, at which point the ore dips 45°, lying nearly on the fault, Fig. 4. The remaining portion lies on the fault and conforms to the fault dip of 35°, Fig. 5.

Open-pit mining for the top of the ore was considered, but since the ore zone dipped into and under the mountains, adverse waste-to-ore ratios precluded use of this method.

The ore occurs in massive quartzite of sufficient strength to support untimbered drifts, crosscuts, and raises. Zones of weakness in the quartzite are bedding, jointing, and small faults or slips. The mineralization, which occurs as small stringers of sphalerite and galena as well as pyrite, creates another line of weakness. The mineral veins or veinlets in themselves are high-grade. Their size and regularity and the amount of barren quartzite by which they are separated determined the limits of mineable ore, which are all *assay limits* except for the one determined by the major fault.

Block 1

Without any background of caving in this type of quartzite, engineers selected the first block on the very steep end of the zone. Compelling reasons prompted this decision. The steep portion of the ore in Block 1 was of the lowest grade, so that if difficulties were encountered no very valuable ore would be lost, while the experience gained might be applied in mining the remaining blocks. A block 200x200 ft was laid out, with four scrams spaced 50 ft apart for drawing and placed at a right angle to the strike. Finger raises were placed in a 25-ft interval grid pattern, with flat undercutting done by crosscuts at the undercut level 25 ft above

The Transactions papers appearing in Mining Engineering during 1953 will subsequently appear in AIME Transactions Vol. 196, and may be permanently referenced to that volume.

the scram. Remaining undercut pillars were blasted out on a diagonal retreat. No timbering was needed at the undercut level. The four scrams were timbered and lined with light rails, and there were six rounds of 10-in. cribbing in the fingers. No boundary weakening was done.

Ore to the mill was to be transported through the main adit level, over 2 miles long. It was anticipated that the caved ore would contain much coarse, blocky material and that underground crushing would therefore be necessary before the caved ore was hauled in mine cars designed primarily to handle the size of ore produced from square sets. A 36x48-in. jaw crusher was installed and set for about 5½-in. discharge size. The crusher is located to serve most of the block caving tonnage, collected by ore passes which eliminate any scraping except in the scrams. A conveyor belt 400 ft long carries crushed ore to the main ore pass, from which trains are loaded on the main mine level 500 ft below.

Scrams of Block 1 were equipped with 20-hp tuggers and 42-in. Eimco folding scrapers. A transfer drift, 10 ft lower than the scrams, provided for transfer to main ore passes by means of a 100-hp tugger and two 60-in. scrapers in tandem. This block was about one-third undercut (15,000 sq ft) when operations were closed by a strike late in August of 1949, and during the three months of the strike no work was carried on here. When work was resumed late in November it was impossible to draw a clean diagonal line at the pillar ends. It was apparent that caving had started in spite of the fact that the undercut area was relatively small. Undercutting of the remaining two-thirds of the block was completed and drawing begun. Despite very low metal prices (Pb 11¢ and Zn 9¾¢) drawing continued until March 1950 to provide necessary operational data. During the four months of drawing 125,000 tons, a record of drawing costs was obtained which confirmed previous estimates. Satisfactory answers were obtained to the important questions: Will the quartzite cave? What will be the size of caved material? It was proved that scram timbers were not subjected to weight. Had this block been operated longer, the rails used to line the scrams would have been replaced with the type of lining now used in the second and third blocks. A solution was still to be found, however, for planning scrams and drawing the orebody as the dip flattened to 65° and subsequently to 35°.

Block 2

The disadvantages of *standard flat undercut* when applied to a dipping orebody are very apparent, resulting in an unwelcome choice. If only the ore were undercut, a considerable wedge of ore up the footwall would be lost even if vertical draw-angle were possible. The alternative, a flat undercut in waste to reach under this footwall ore, would necessitate drawing large amounts of waste to reach ore.

It was decided to undercut and draw by means of scrams parallel to and under the ore footwall, an operation later discovered to be strikingly similar to that used by the Mufilira Copper Co. in Africa. This plan involved a stair-step pattern of scrams up the footwall, obviously risking the unknown effect of subsidence, since even a minor deviation, or *bulge*, in draw angle for drawing in a lower scram would destroy the scram above it.

The stringers of mineralization, and some shearing, were steep and somewhat parallel to the proposed plane of the scrams. If these mineralized veins or the shearing were too weak, slippage along them would destroy the scrams. However, bedding and jointing of the quartzite, which is generally at right angles to the steep veins, proved to be of equal or greater weakness. At the end of 1952, with total production of 591,000 tons from Block 2, no subsidence or slipping has occurred in the plane of these footwall scrams.

Undercutting is accomplished by shrinkage on the ore footwall. After the scram is finished, finger raises are driven, connected, and belled out. Shrinking begins and undercuts the ore footwall until the finger raises of the next successively higher scram are reached. Access to the shrink is from small temporary raises driven from the shrink bottom to hole out in the side of the scram above. Ultimately these temporary raises become part of the shrink as work progresses upward. This undercut method has been used successfully on the ore footwall where the dip is not less than 45°. Subsequent undercutting in future blocks with less than 45° dip will require some modification. However, the major fault, which is very weak, adjoins much broken ground, so that undercutting may not be required. The problem may well be to get finger raises in and belled before this fault starts to *run* up dip.

Each of these inclined blocks is served by an orepass and service raise. Block 2 was 200 ft long, measured horizontally along the footwall. Block 3 will be 300 ft long with the orepass and service raise in the center to serve 150 ft in each direction.

Scrams

Of particular interest is the evolution of scram linings. Rails have been tried in Block 1, but failed on impact of either the heavy scraper or large boulders *paying-off* from finger raises. Lighter scrapers of less capacity would result in fewer tons per manshift. Scrams were therefore modified to accommodate the Eimco folding scraper, which had proved very adaptable in mining Bunker Hill ore.

Scram sets are 12x14-in. timber, the posts 7 ft 8 in. long and the caps 8 ft long. Usually these sets are placed on 6-ft centers. End-grain blocks are placed between the sets and a shear wall is created which takes tremendous abuse. To protect the scram posts, blocks are about 2 in. longer than the post is thick, see Fig. 6. In addition to this, scram posts and caps are bolted together with threaded rods on the finger raise side of the scram. This practice helps to prevent finger posts being knocked out and holds end-grain blocks tight in sets immediately adjacent to the open set where the finger is located, see Fig. 7. To be successful, these end-grain blocks must be put in very tightly, with filling behind them if there are irregularities in the scram.

This type of scram has provided a lining which has cut repairs to a bare minimum. Once these end-grain blocks are installed properly no maintenance is required as the relatively coarse caved ore is drawn, see Fig. 8. No lining is used on the bottom of the scram. Scram posts are battered about 6 in. out at bottom. This has been sufficient to keep posts from *kicking in*. Grizzles, at 2-ft centers, control size of material discharged to ore passes, Fig. 9.

The dimensions of the finger set are 6x5-ft and the maximum dimensions of irregular boulders handled by scraping are 3x3x2½-ft.

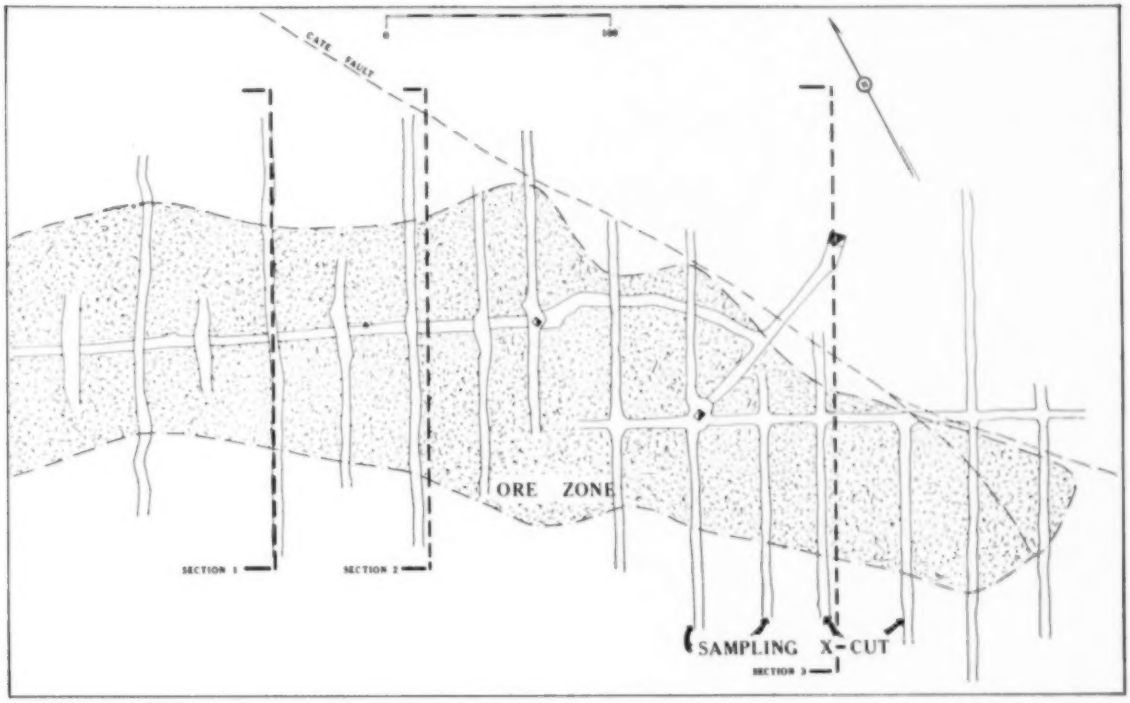


Fig. 1—Plan view of Bunker Hill orebody, 1000 ft long and 165 ft wide in horizontal section. The vertical extension is about 300 ft from an old level to the surface.

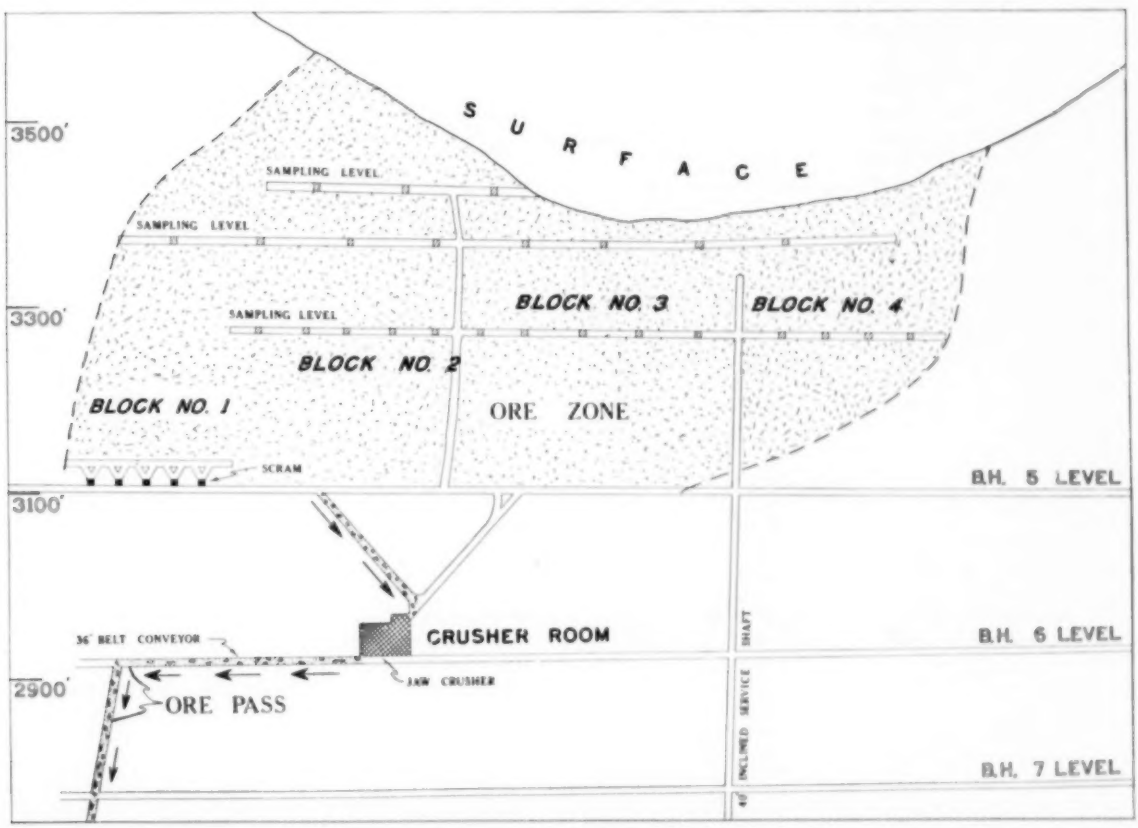


Fig. 2—The longitudinal projection for mining a section of the Bunker Hill orebody shown in Fig. 1.

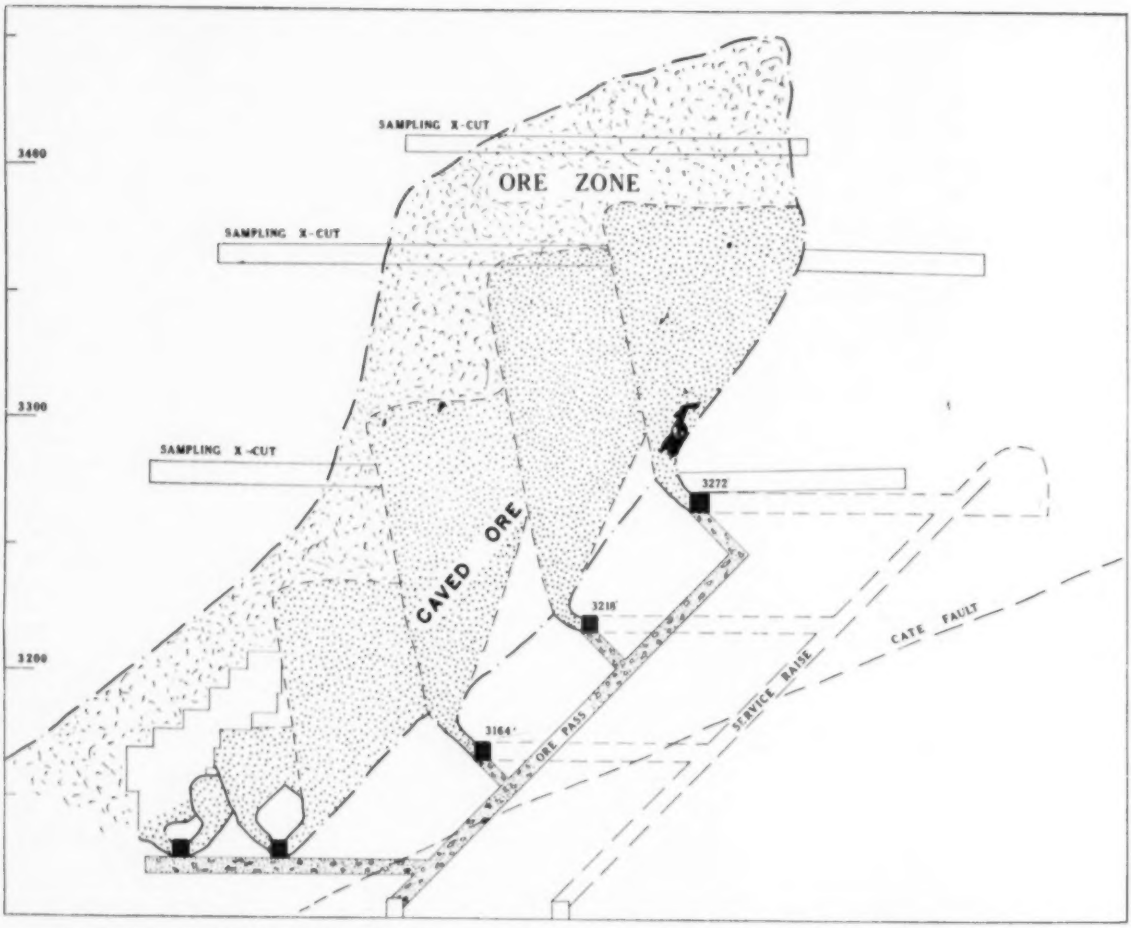


Fig. 3—Section 1 of the Bunker Hill orebody. The ore lies in the hanging wall of a major fault of the Bunker Hill mine. The wedge of waste which separates the ore from the fault pinches out near the center of the zone, where the ore dips 45°, lying nearly on the fault as shown in Fig. 4.

When coarseness of the caved ore and the relatively low daily production for a block-caving program are taken into account, this type of scram construction with 20-hp tuggers and the Eimco folding scraper has been the optimum combination from a standpoint of drawing costs and maintenance costs. There has been little wear on the end-grain blocks, and most of them can be used again after drawing is completed.

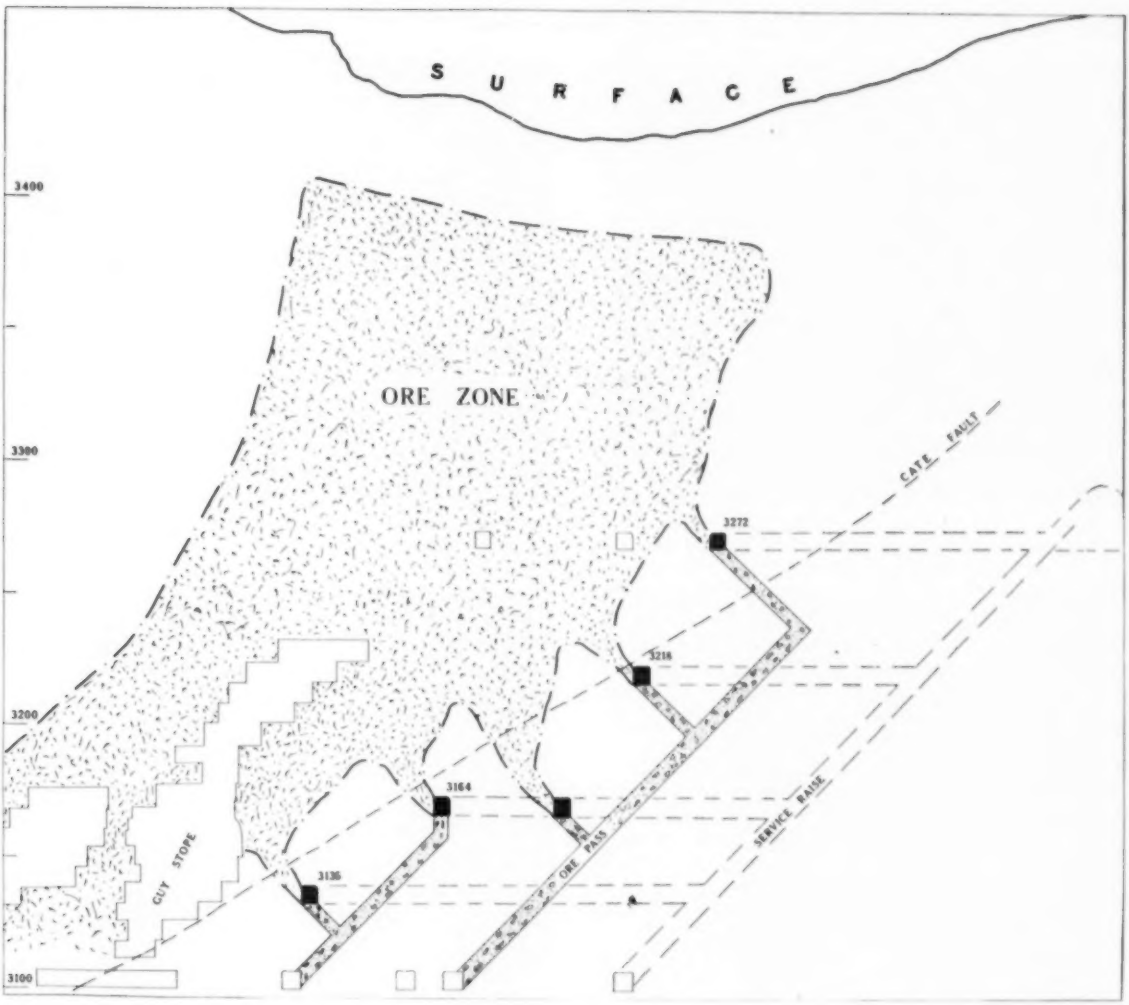
Concrete linings have been considered and may be used experimentally in future. The economic advantage of using concrete on the Bunker Hill job is doubtful, however, particularly if end-grain blocks can be used in two or three locations before being discarded.

Ore passes have also been lined with end-grain blocks where the condition of the ground demanded lining to control wear. These ore passes are generally lined with longer blocks, from 18 to 24 in., depending on the amount of rough, coarse ore that is to be handled.

Drawing

Draw control is used to maintain *ore-hangwall and waste* contact at about the same angle as it was originally. Rate of draw, measured horizontally, equals ½ ft per day. If fingers do not *hang up* when drawn sufficiently, a few pieces of lagging will shut them off. Secondary blasting consumes 0.08 lb of powder per ton of ore and blasting high hang-ups is done by means of dynamite tied to blasting slats. When hang-ups are more than 20 ft from the fingers, which happens rarely, they are blasted from the scram so that miners will have an opportunity to get away if the finger runs before anticipated. Ventilation to each scram comes from the service raises which are under pressure. Fresh air sweeps past the tugger and exhausts up small ventilation raises located at scram ends. The tugger man works in the fresh air stream and dust and smoke are driven away from him. High pressure water sprays are also used to minimize dust caused by scraping. The 8000 to 10,000 cu ft of air forced into each scram clear powder smoke of secondary blasting without much loss of production time. The single man who works in these producing scrams draws an average 150 tons of ore per shift while doing all necessary blasting and making minor repairs.

At the time Block 2 was 40 pct drawn, undercutting of the adjacent end of Block 3 was begun and put under slow initial draw. The undercutting of Block 3 will continue at a rate calculated to bring in new fingers as those at the extreme end of Block 2 are finished. This results in a slow retreat across the whole ore zone and attempts to establish and maintain a broken ore-broken waste contact or plane which gradually moves along the strike of the ore at a constant angle. The angle of subsidence, or



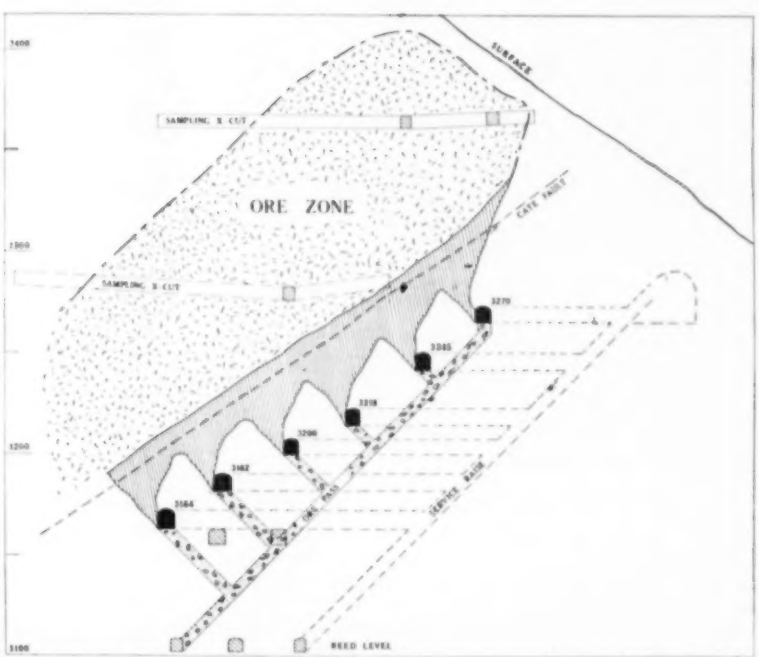


Fig. 4 (above)—Section 2 of the orebody. The wedge of waste which separates the ore from the fault pinches out in this section, where the ore dips 45°, lying nearly on the fault. Fig. 5 (right)—Section 3 of the orebody.

Re-usable End-Grain Wood Blocks Protect Scram Walls

Fig. 6 (left)—To protect the scram posts, blocks are set about 2 in. longer than the posts are thick. Scram posts and caps are bolted together with threaded rods on the finger raise side of the scram. Fig. 7 (right)—End grain blocks must be put in very tightly, with filling behind them if there are irregularities in the scram.

Fig. 8 (left)—As shown above, caved ore is comparatively coarse. Fig. 9 (right)—Grizzles at 2-ft centers control the size of the material discharged to the ore passes.

angle of draw, has been measured as 82° to 83° inclined into the block.

No calculations of drawing efficiency are available at this time, since no blocks are completely drawn. Also, such calculations are dependent on metal prices. For instance, Block 1 at high metal prices still contains ore, and since there is no weight on the scrams, its openings are still in good condition ready for additional drawing if market conditions warrant it. If the scrams had taken weight, as normally occurs in many block-caving operations, it would have been impossible to maintain economical operations with the violent fluctuations in lead and zinc prices in the last four years.

For the first 10 months of 1952 the total cost per ton was $3.97 for all mining, transportation, and milling. This figure covers all chargeable expense, excluding only general plant depreciation and corporate income taxes. Although today's very low lead and zinc prices forced curtailment of work February 7, at reasonable market prices these operations are highly successful. There is little doubt that they have played some part in the evolution of a method of block-caving peculiarly adaptable to dipping orebodies, and it is almost a certainty that this first orebody is a fair sample of the type of occurrence to be expected in other areas at Bunker Hill mine.

In this presentation it has been possible to discuss only the general facts concerning block caving at Bunker Hill mine. Inquiries are invited from those interested in additional data.

Alkali Reactivity of Natural Aggregates In Western United States

by William Y. Holland and Roger H. Cook

In view of the increasingly widespread deterioration of concrete structures as the result of the interaction of the alkalies sodium and potassium released by hydration of portland cement and susceptible rocks and minerals in aggregates, it is believed that a paper summarizing the geographic distribution of these aggregates will be of interest to producers and users of concrete, concrete products, and concrete-making materials.

THIS paper reviews the problem of alkali-aggregate reaction in concrete and describes the geologic and geographic occurrence and distribution of alkali-reactive sand and gravel in western United States. It includes no discussion of crushed stone or synthetic aggregates.

Most deposits of sand and gravel are accumulations of particles of rocks and minerals from a variety of sources, and it is not unusual for at least one or two varieties of the rocks to contain some form of reactive material. Examination by petrographic methods of many sands and gravels, as well as manufactured aggregates, has shown that a comparatively high proportion of the deposits does contain, in greater or lesser degree, rocks and minerals known to be deleteriously reactive with the alkalies of cement. Fortunately the amount of reactive materials is commonly less than that necessary to cause deleterious effects in concrete.

As investigation of unsound concrete structures progresses, it becomes evident that the alkali-aggregate reaction is even more widespread than supposed, Figs. 1 and 2. Even though some parts of the country appear at present to be immune, further investigations will probably show that the effects of alkali-aggregate reaction can be seen in many structures in these areas, although only on a small scale in most of them. Many concrete structures will, of course, have lived their useful life before disintegration from this cause is serious, and in others the alkali-aggregate reaction may never become significant even though the microscopic evidence of reaction is present.

The alkali-aggregate reaction first was reported to be a cause of deterioration of concrete in 1940 when Stanton[1] described expansion of concrete pavements in California. Similar expansion and deterioration of concrete was recognized during succeeding years in concrete structures located in many parts of the country, but particularly in the western states. A number of concrete laboratories[2] became interested in the problem. It was soon determined that only certain combinations of aggregate and cement caused the alkali-aggregate reaction to take place, and moreover that the reaction progresses only in the presence of water. Further research proved that cements containing more than 0.60 pct total alkalies (pct Na_2O + 0.658 x pct K_2O), when used with aggregates containing appreciable amounts of reactive ingredients, caused the reaction to take place, usually with subsequent deterioration of the concrete.

During the last few years this limitation has been adhered to in both government and private construction as the maximum allowable alkali content of cement to be used with aggregates of known alkali reactivity. Because of this limitation, it appears that deleterious reaction either has been reduced or eliminated in many recently built structures in which it probably would otherwise have occurred.

Recent tests have shown that the degree of expansion obtained with any particular cement-aggregate combination depends not only on the alkali content of the cement but also upon the relation of this alkali content to the amount and degree of reactivity of reactive constituents in the aggregate.[3] In laboratory mortar bars, opal and cements with alkali content of as low as 0.2 pct (as equivalent of Na_2O) have produced deleterious expansion as the result of alkali-aggregate reaction. These experiments demonstrate that aggregates containing even 0.1 pct of opal are deleteriously reactive.

It was soon determined that alkali-silica gels were formed by the interaction of the alkalies of the cement and the reactive aggregate, Figs. 3 and 4. Osmotic or swelling pressures produced by the continued hydration of these gels cause expansion of the concrete with resulting cracking, warping, and dislocation.

Evidence of the alkali-aggregate reaction can be seen by a petrographic study of the deteriorated concrete. Among the first structures studied by this method was Parker Dam on the Colorado River, California-Arizona. In the concrete from this dam pebbles of rhyolite, andesite, siliceous limestone, and chalcedonic chert were found to be reactive.

W. Y. HOLLAND and R. H. COOK are Petrographers, Engineering Laboratories Branch, Design and Construction Division, Bureau of Reclamation, Denver, Colo.

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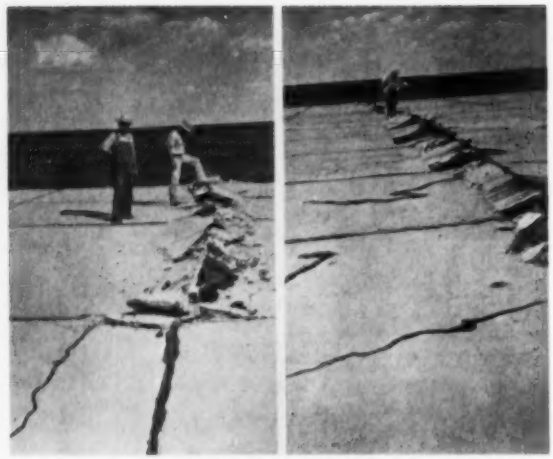


Fig. 1—McCook air base, Nebraska. Expansion of concrete has resulted in extreme buckling at expansion joints.

Reaction appeared to be progressive, characteristically developing a rim extending from the exterior of the pebbles toward the interior. In many examples the whole pebble appeared to have reacted, in others only a rim of varying thickness. On close examination the gels formed by the interaction of alkalies of the cement and the volcanic glass, opal, and chalcedony of these rocks could be seen in the immediately surrounding mortar, within the aggregate, and in microfractures radiating from the reacted pebbles, Figs. 5 and 6. Similar microscopical evidence of alkali-aggregate reaction has been found in association with other known reactive materials.

Ameliorative Measures

Considerable progress has been made toward the development of materials and methods that retard or even eliminate the alkali-aggregate reaction. One of the first ameliorative measures taken was to limit the amount of alkalies in cements that are to be used with aggregates of known reactivity. It has been found that the use of various pozzolans such as the opaline shales and cherts of the Monterey formation of southern California will retard the expansion. Air-entraining agents in certain proportions also reduce the expansion resulting from this action. With air-entraining agents, the chemical reaction may progress, but the products of the reaction (the gels) probably find room for their volume increase in the air voids. The addition of small amounts of certain materials such as lithium carbonate and some proteins to high alkali cement have an inhibiting effect on the expansion reaction.[4]

Alkali-Reactive Materials

Identification of reactive materials in an aggregate can be made petrographically. Such examinations are now routine in many concrete laboratories. A competent petrographer can make the examination in a comparatively short time.[5,6] The usual procedure is to examine and identify the pebbles from a sample of each fraction of gravel and sand under a binocular microscope. If more detailed observation is necessary for the identification, petrographic thin sections or mounts of crushed particles are examined under a petrographic microscope. Each fraction is separated into its constituent lithologic groups on the basis of this examination and the percentages are calculated. If reactive ingredients are disclosed, further study is made to establish the amount and probable degree of reactivity of these constituents.

Aggregates now known to be deleteriously reactive contain one or more of the following, as shown also in Table I: Acidic to intermediate volcanic rock (e.g., glassy to cryptocrystalline rhyolites, latites, dacites, andesites and their tuffs, and glassy rocks such as obsidian, perlite, and pumice); the silica minerals opal, chalcedony, tridymite, and cristobalite; the zeolite heulandite, and probably ptilolite and certain phyllites. Any rock containing a significant proportion of reactive substances will be deleteriously reactive; thus, although pure limestone and dolomites are not reactive, opaline or chalcedonic limestone and dolomites are deleteriously reactive. Similarly, any normally innocuous rock can become deleteriously reactive if coated or impregnated with opal, chalcedony, or other reactive substances.

Based on investigation of aggregates of known deleteriousness, estimates can be made of proportions of various constituents necessary to produce distress in concrete. Aggregates containing more than 0.25 pct by weight of opal, more than 5 pct of chalcedony, or more than 3 pct of glassy or cryptocrystalline acidic to intermediate volcanic rocks are regarded by the Bureau of Reclamation as harmfully reactive. These limits are probably conservative; to the extent that reactive ingredients are admixed or enclosed within innocuous substances the limits can be increased. For example, opal may occur in isolated masses within innocuous material and will be less susceptible to alkali attack.

The acidic to intermediate volcanic rocks owe their reactivity primarily to the natural volcanic glass, Fig. 7, of which they are wholly or in part composed. Volcanic rock may range from a wholly glassy material, such as perlite and obsidian, to a fine-grained porphyritic variety, such as andesite porphyry. The extent to which a volcanic rock will contribute to alkali reactivity depends on the amount of natural glass or cryptocrystalline material in its ground mass. Particles of a reactive volcanic rock type may be only slightly deleterious or even innocuous if the original glass is completely recrystallized as a result of geologic processes. The natural glasses are an amorphous substance usually containing from 70 to 75 pct silica in the acidic or rhyolitic varieties and from 50 to 65 pct in the intermediate or andesitic varieties. Basaltic glasses containing 45 to 50 pct silica are not deleteriously reactive with cement alkalies. Microscopical observations over extended periods of samples of glassy tuffs mixed as a slurry in alkaline solutions, such as occur with hydrating high-alkali cements, have shown that the glass slowly reacts with, or dissolves in, the alkaline solutions and forms alkalic-silica gels.

Opal, opaline cherts and shales, and other rocks containing opal owe their reactivity to the amorphous, hydrous silica (opal) of which they are composed wholly or in part, Fig. 8. It has been shown that amorphous hydrous silica combines readily with the alkaline solutions of cement to from gels which are clear at first but ultimately turn white.

Rock containing certain zeolite minerals can contribute to alkali-aggregate reaction indirectly through cation-exchange reactions of the zeolites.

Sodium and potassium which are thus released augment the supply of alkalies from the cement. Tests have been made on analcite-bearing phonolite porphyries from Missouri Buttes and Barlow Canyon, Wyo., and on a basalt from Table Mountain, near Golden, Colo., which contain several zeolite minerals, including analcite, chabazite, and natrolite. These tests indicate that alkalies are released by cation-exchange reactions between the zeolites and calcic solutions produced by the hydrating portland cement. Examination of mortar bars fabricated with these materials indicated that the rocks themselves do not cause deleterious expansion if used as the entire aggregate, even with high-alkali cement. On the other hand, when ground and used as an addition or partial replacement of low-alkali cement in mortar containing alkali-reactive aggregate, the rocks release sufficient sodium and potassium to cause significant expansion of mortar.[9]

Recent studies[10] with the electron microscope have shown that chalcedony probably consists of fine-grained quartz and a variable quantity of free water; no opal could be identified. If chalcedony is composed essentially of fine-grained quartz, its reactivity must result from the extremely high surface area available for attack by alkalies. It is known that pulverized quartz releases considerable silica to alkaline solutions, whereas massive quartz releases comparatively little. Moreover, recent investigations[11] show that particles of quartz dust possess a highly soluble surface layer and that this layer can be removed easily by various solvents with the formation of a flocculent colloidal silica.

Fig. 2—Example of alkali-aggregate reaction, Coolidge Dam, Arizona. Crack in pier on left end of upstream parapet wall. Maximum opening of crack, 1½ in.

Much has yet to be learned about the reactivity of the other minerals mentioned in Table I. However, they undoubtedly perform deleteriously in a manner similar to the natural glasses and amorphous hydrous silica, that is, as a consequence of attack by alkaline solutions, they produce a gelatinous substance that increases in volume by imbibition of water.

Table I. Alkali-Reactive Rock and Minerals

REACTIVE MINERALS	COMPOSITION
Opal	$SiO_2 \cdot nH_2O$
Chalcedony	SiO_2
Tridymite[7]	SiO_2
Cristobalite[8]	SiO_2
Heulandite	$H_4CaAl_2(SiO_3)_6 \cdot 3H_2O$
REACTIVE ROCKS	
Siliceous Rocks	
Opaline cherts	Opal
Chalcedonic cherts	Chalcedony
Siliceous limestones	Chalcedony and/or opal
Volcanic Rocks	
Obsidian, perlite, pumice	Acidic to intermediate volcanic glass
Rhyolites and rhyolite tuffs	
Latites and latite tuffs	
Dacites and dacite tuffs	
Andesites and andesite tuffs	
Miscellaneous Rocks	
Any rocks containing veinlets, inclusions, or detrital grains of the reactive substances listed above.	

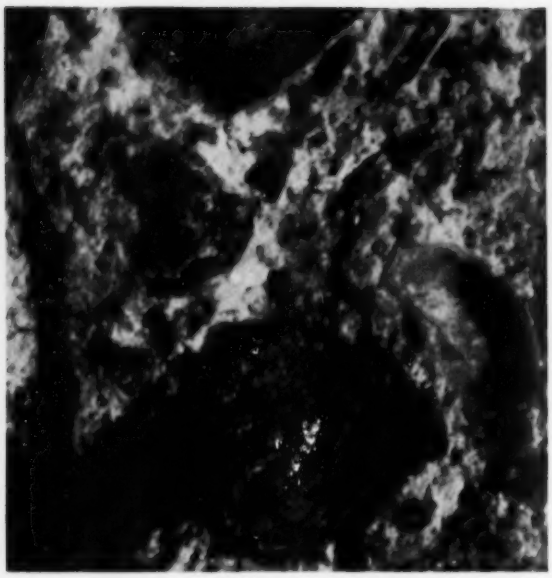
Alkali-Reactive Aggregate Deposits

Although alkali-aggregate reaction has been identified in concrete structures in the eastern and southeastern parts of the United States, alkali-reactive aggregate is of particular concern to users and producers of concrete and concrete products in the western states because of the widespread occurrence of potentially reactive rocks and minerals. The map, Fig. 9, shows the geographic distribution of some 245 deposits which were investigated in the Bureau of Reclamation laboratories and found to contain deleterious proportions of reactive rock types. Most of these deposits are located in the drainage basins of the Missouri, Platte, Rio Grande, Arkansas, Columbia, Snake, Colorado, Sacramento, and San Joaquin Rivers.

Only deposits of reactive aggregates are shown on the map. However, this is not intended to imply that all aggregate deposits in these areas are reactive. On the contrary, of the many additional aggregate deposits investigated in these river basins, some were composed wholly of innocuous rock types; others were found to contain reactive rock types in amounts less than those considered necessary to produce a harmful degree of reaction in concrete.

The lithologic composition of a specific aggregate deposit is the result of a complex geologic history. Deposits of natural gravel and sand have their origin, first, in the weathering and erosion of rock formations. Secondly, the particles of ledge rock produced by erosion are transported from their point of origin to their present location. The major transporting agencies, in the case of natural aggregate suitable for use in concrete, are streams and glacial ice.

The lithologic composition of gravel and sand deposits is controlled by the rock types available at the sources. If formations composed of potentially reactive rock, such as a glassy rhyolite lava flow or a siliceous limestone, occur within the drainage basin of a river, it is inevitable that at least



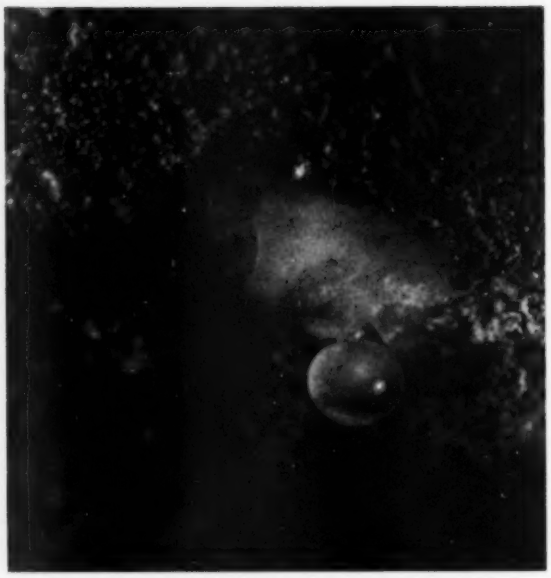


Fig. 3 (left)—Photomicrograph shows partially decomposed pebble, gel-lined void, and accompanying gel in concrete. From a highway in California. Original magnification X25. Fig. 4 (right)—Photomicrograph of alkali-silica gel bead extruded from laboratory concrete. Original magnification X6.

some of the reactive rock will be eroded and find its way into aggregate sources along the river. The proportion of reactive rock types in an aggregate will depend, at least in part, on the relative abundance of reactive rock formations in contrast to nonreactive rock formations, as well as the relative ease with which each type of rock can be eroded, the kind and severity of the transporting processes involved, and the distance from the source.

Tributaries to major streams contribute to local variations in kind and amount of rock types occurring along the main streams and may thus influence the reactivity of the aggregates. If, for example, a tributary flows through an area composed predominantly of potentially reactive andesitic rock, the aggregate along the tributary will, of course, be predominantly andesitic and reactive. Hence the concentration of reactive andesites in sand and gravel carried by the main stream will be increased below the confluence of the streams. The seasonal variations in the amount of sediment transported by streams may repeatedly alter the lithology of deposits.

Terrace deposits frequently present special problems. Terrace deposits of aggregates deposited by ancient streams and glaciers are often eroded and redeposited as a result of changes in course of present-day streams; deposits so formed will differ lithologically from deposits along the same stream formed from bedrock material. Sand and gravel in

Fig. 5 (left)—Photomicrograph of gel-lined void in thin section of concrete from Stewart Mountain Dam, Arizona. Ordinary light. Original magnification X120. Fig. 6 (right)—Photomicrograph of gel-filled crack in thin section of concrete from Stewart Mountain Dam. Ordinary light. Original magnification X120.

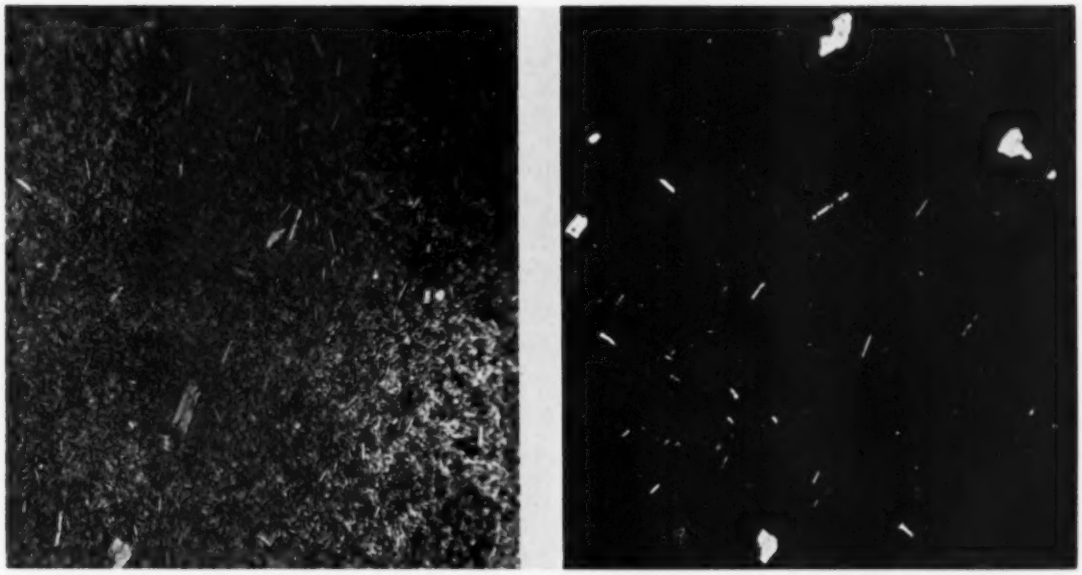


Fig. 7—Photomicrographs of a thin section of potentially alkali-reactive glassy rhyolite. Ordinary light on left and crossed nicols right. The dark phase appearing in the right photomicrograph is the glass constituent. Original magnification X120.

terrace deposits may be impregnated or coated with opaline or chalcedonic material brought in by percolating water; the aggregates from these deposits thus would be alkali-reactive, even though the area is free from formations composed of alkali-reactive rocks. Some terrace deposits may contain volcanic or other reactive rock types derived from ancient and now nonexistent sources and hence will differ from sand and gravel deposited along the existing channel.

Fundamentally, for alkali-reactive rocks to be present in an aggregate deposit, formations capable of supplying the reactive rocks and minerals must be present in the source areas for that deposit. In the 17 western states such potential source areas of reactive materials are abundant. Geographic distribution of volcanic lava flows in the western states is shown on the map, Fig. 10. The shaded areas indicate volcanic rock flows in which rhyolites, trachytes, latites, dacites, andesites, and their related tuffs are reported to occur. It is not intended to imply that all the rocks in the areas shown have been proved to be capable of producing harmful reactions with cement alkalies. The potential deleteriousness of the rock in each of these areas would have to be ascertained individually by field and laboratory investigations. However, from the petrographic nature of these rocks the probability is high that glassy or cryptocrystalline phases capable of reactivity are present.

During the Miocene period of geologic time most of the Columbia River plateaus, which include the

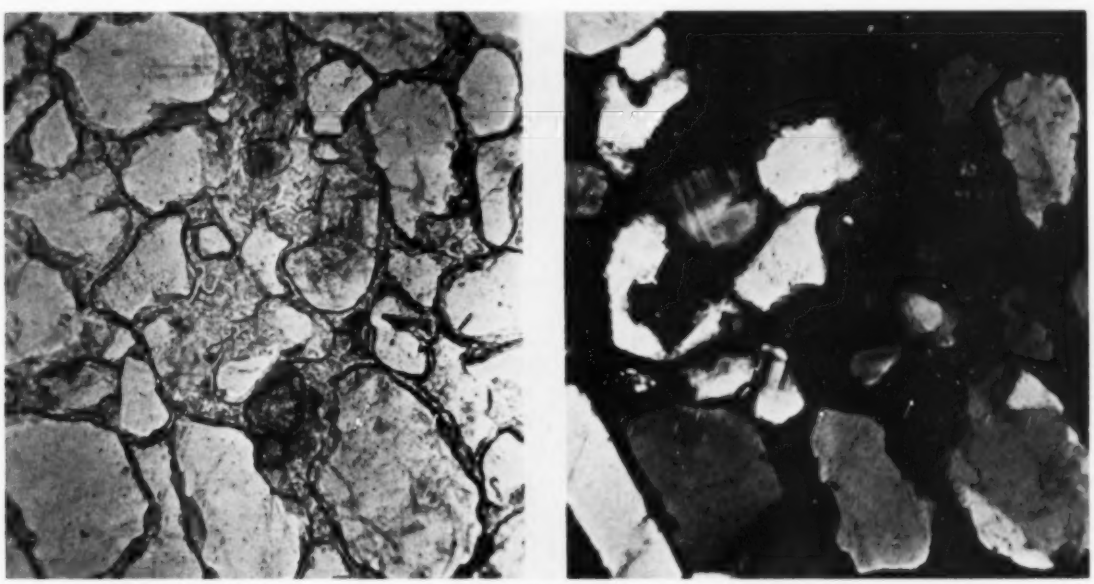


Fig. 8—Photomicrographs of a thin section of potentially alkali-reactive opaline sandstone. Ordinary light on left and crossed nicols right. The dark phase between the sand grains in the photomicrograph on the right is the opaline material. Original magnification X120.

Fig. 9— Alkali-reactive aggregate deposits in western United States.[12]

central and eastern portions of Washington and Oregon and the western edge and southern portion of Idaho, were covered by thick flows of basalt. Consequently, the aggregate deposits in this province are composed predominantly of basalts and with rare exceptions are not reactive, unless as in some of the deposits in the Columbia Basin area of central Washington, the basalt particles are heavily coated with opal. Much smaller flows of andesite and rhyolite have occurred during earlier and later geologic periods in several localities within this province. These flows contribute alkali-reactive materials to local deposits. For example, the Deschutes River in central Oregon has cut through the Columbia River basalt, exposing the Clarno formation of lower Oligocene and upper Eocene age. This formation, locally as much as 2000 ft thick, is composed of aphanitic to glassy andesites, glassy rhyolites, and perlites, with interbedded layers of tuffs, ash, and breccia. Results of alkali-aggregate reaction have been recognized in concrete of a highway bridge, Fig. 11, across the Deschutes River near Redmond, Ore.

Andesitic to rhyolitic volcanic rock flows are scattered throughout the Pacific Coast province, which includes the western portions of Washington and Oregon and most of California. In addition to the volcanic rock, sedimentary formations, such as the Franciscan and Monterey formations in California and the Oligocene and Miocene formations along the coasts of Washington and Oregon, have contributed opaline and chalcedonic rock to the various aggregate deposits.

Evidence of alkali-aggregate reaction has been recognized in concrete, Fig. 11, at the Hetch Hetchy power plant, in a railroad bridge near Friant Dam, in highway structures near Fresno in the San Joaquin Valley, in a flume structure in the southern Sierra Nevada, in buildings and structures in and near Los Angeles, in highway structures and sea walls at and near Santa Barbara and Ventura, and in highway structures near Bradley and Paso Robles in the Salinas Valley.

Tertiary volcanic activity was widespread in the Great Basin geologic province, which includes the eastern edge and southeastern portions of California, all of Nevada, the western edge of Utah, the western edge and southern half of Arizona, and the southwest corner of New Mexico. Flows of acidic to intermediate volcanic rock are abundant in this area, some of which contributed reactive rhyolites, dacites, andesites, and trachytes to the aggregate deposits along the lower Colorado River. Particles of chalcedonic cherts and siliceous limestones contributed by various sedimentary formations in the area are found in the Colorado River aggregate deposits. Aggregates from terrace deposits along the lower Colorado River were used in concrete at Hoover and Davis Dams and Parker Powerhouse, Fig. 11, constructed by the Bureau of Reclamation, and at the Gene wash and intake pumping plants of the Metropolitan Water District of southern California. Aggregates used in the construction of Parker and Copper Basin dams were obtained from a terrace deposit along the Bill Williams River, near its junction with the Colorado River. Expansion in excess of 0.1 pct linearly has been measured in concrete at Parker Dam. Gene Wash and Copper Basin dams are affected by alkali-aggregate reaction. The aggregates used at Davis Dam proved to be deleterious in laboratory tests of concrete and mortar containing high-alkali cement. Petrographic analysis of aggregates used at Davis Dam identify the reactive constituents as rhyolites, dacites, andesites, chalcedonic cherts, and siliceous limestones. These rock types constitute about 45 pct of the analyzed gravels. Pozzolanic material was used in construction of Davis Dam to relieve this condition.

The Rocky Mountain and Colorado Plateau province, which includes the eastern part of Idaho, western Montana, Wyoming, and Colorado, northwestern New Mexico, northern Arizona, and eastern Utah, contains numerous flows of acidic and intermediate volcanic rock. Such flows along the Continental Divide in northwestern Wyoming, southwestern Montana, and eastern Idaho, have contributed potentially reactive rock types to the drainage basins of both the Missouri and Snake Rivers. An aggregate considered for use at Blue Mesa Dam in

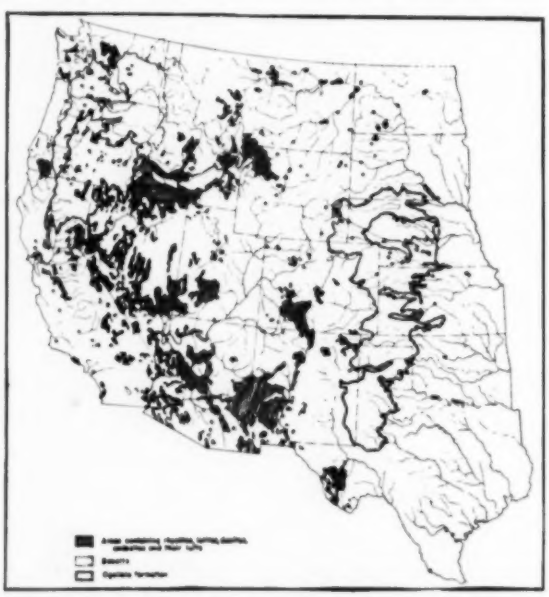


Fig. 10—Distribution of alkali-reactive aggregate deposits and lava flows in western United States.[12]

Fig. 11—Alkali-reactive aggregate deposits (small dots) and location of structures (large dots) that have been affected by alkali-aggregate reaction in western United States.[12]

western Colorado contains from 27 to 50 pct of rhyolites, andesites, tuffs, and chalcedonic sandstones. Mortar bars made from this aggregate and high-alkali cement show expansions of up to 0.473 pct in 12 months. Sedimentary formations containing reactive material are common in the area, such as the Bozeman formation in southwestern Montana, which contains rhyolites and opaline and chalcedonic rocks. Opaline and chalcedonic sandstone and limestones are found in southwestern Wyoming. The Marble Canyon limestones in northern Arizona contains chalcedonic zones.

Volcanic activity is virtually absent in the Great Plains province which includes eastern Montana, Wyoming, Colorado, and New Mexico, western Oklahoma, and Texas. However, the area is covered with large sedimentary formations, such as the Ogallala, which contain potentially reactive rocks and minerals. A portion of the Ogallala formation is composed of an opaline sandstone which contains up to 40 pct of opal as cementing material. Since many of the rivers in this area have their headwaters in the Rocky Mountain region, some volcanic rocks have been carried into this area. Volcanic glass particles present in some formations in this area probably are the result of explosive volcanic activity in the Rocky Mountains, particularly from the Yeliowstone Park area. Evidence of alkali-aggregate reactions has been found in concrete in pavements and curbings at Kimball, Neb.; air-strip pavements at McCook, Neb.; and in highway and other structures at Lincoln and Gretna, Neb., and Saint Francis, Norton, Clay Center, and Concordia, Kan., see Fig. 11. The aggregates used in these structures were obtained from the Platte and Republican Rivers or their tributaries.

New evidence of alkali-aggregate reaction continues to be found in old concrete structures. Only recently two large 40-year-old bridge slabs in western Missouri were examined and found to contain effects of the reaction. In this case the alkali-aggregate reaction was augmented by freezing and thawing. This concrete contained chert aggregate which, as previously mentioned, has been found to be reactive. Studies of affected structures and continued research on ameliorative measures to control the reaction are in progress and will help eliminate this troublesome factor in concrete engineering.

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Corrections

In the June 1953 issue: TP 3554A. Testing of Roof-Bolting Systems Installed in Concrete Beams. By Rudolph G. Wuerker. On p. 608, col. 1, Abram's formula should read:

$$S_{compr.} = 7 \frac{\frac{14,000}{w}}{c}$$

In the July 1953 issue: TP 3544F. The Blending of Western Coals for Production of Metallurgical Coke. By John D. Price. P. 718, Fig. 2, caption should read: Effect of blending coals of similar rank. Colorado high-volatile coals. Fig. 3, caption, should read: Effect of blending high and low-volatile coking coals. Colorado high-volatile coal, above, and Utah high-volatile coal, below. P. 719, Fig. 4, caption, should read: Effect of use of char. Colorado high-volatile coal, above, and Utah high-volatile coals, below. Fig. 5, caption, should read: Effect of use of coal tar pitch. Colorado high-volatile coal, above, and Utah high-volatile coal, below. P. 720, Fig. 6, caption, should read: Effect of use of petroleum coke. Colorado coal plus petroleum coke.

Orientation of Cube Diamonds in Drill Bits

by Eugene P. Pfleider

Research indicates that the cube diamond is a superior stone for drill bits. It can be oriented to combine a long cutting edge with structural strength and high resistance to wear. Test bits set with preferred orientation give double the bit life at astoundingly low diamond loss in comparison to oriented octahedral diamonds.

A PROGRAM of research on the orientation of diamonds in drill bits has been under way for several years at both the United States Bureau of Mines Experimental Mine at Mount Weather, Va., and the School of Mines and Metallurgy of the University of Minnesota. Reports on past work done both in the laboratory and the field are available in the technical press.[1, 2, 3] The results of such drilling, done under comparable conditions, show that bits having diamonds oriented in a direction that combines structural strength and greatest hardness not only can advance somewhat faster than random set bits but can do this with a unit diamond loss of 50 pct or less. Several bit manufacturers have initiated the training of diamond setters in the principles of orientation on the basis of the above-mentioned research results.

The basic information on the crystallography and hardness variation of the diamond was presented in an earlier paper,[1] which pointed out that the diamond is several hundred times harder along certain directions and certain faces than on others. The same article discussed the drilling results obtained with a group of nine bits having octahedron diamonds set with *on-point*, positive *(gouge)* rake, and negative *(drag)* rake orientations. The testing of oriented cube diamonds is an extension of this research program, employing the same procedures and equipment (Hy Mac hydraulic drill unit, EXT diamond core bits, St. Cloud Pink granite) as previously described.

This current work covers the test experience of seven bits set with cube diamonds running approximately 10 stones per carat. The individual stones were near perfect cubic form; hence it was simple to orient them in any desired direction. The cube is a comparatively rare crystal form of the diamond, although considerable numbers are produced from the Congo. It is understood that most of the cube diamonds are retained in Africa, where they are either used locally for diamond bit manufacture or ground up to make powder. They have been considered to be an inferior stone for drilling purposes.

E. P. PFLEIDER, Member AIME, is Chairman, Department of Mining Engineering, School of Mines and Metallurgy, University of Minnesota, Minneapolis.

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Fig. 1 (left)—Experimental drill bit M-15 having a point orientation with a positive (gouge) rake off cube face. Apparent rotation, counter clockwise. Fig. 2 (center)—Bit M-14 having a point orientation and a negative (drag) rake off cube face. Rotation counter clockwise. Fig. 3 (right)—Bit M-16 having edge orientation and a positive rake (20° to 25° gouge angle) off cube face. Apparent rotation counter clockwise.

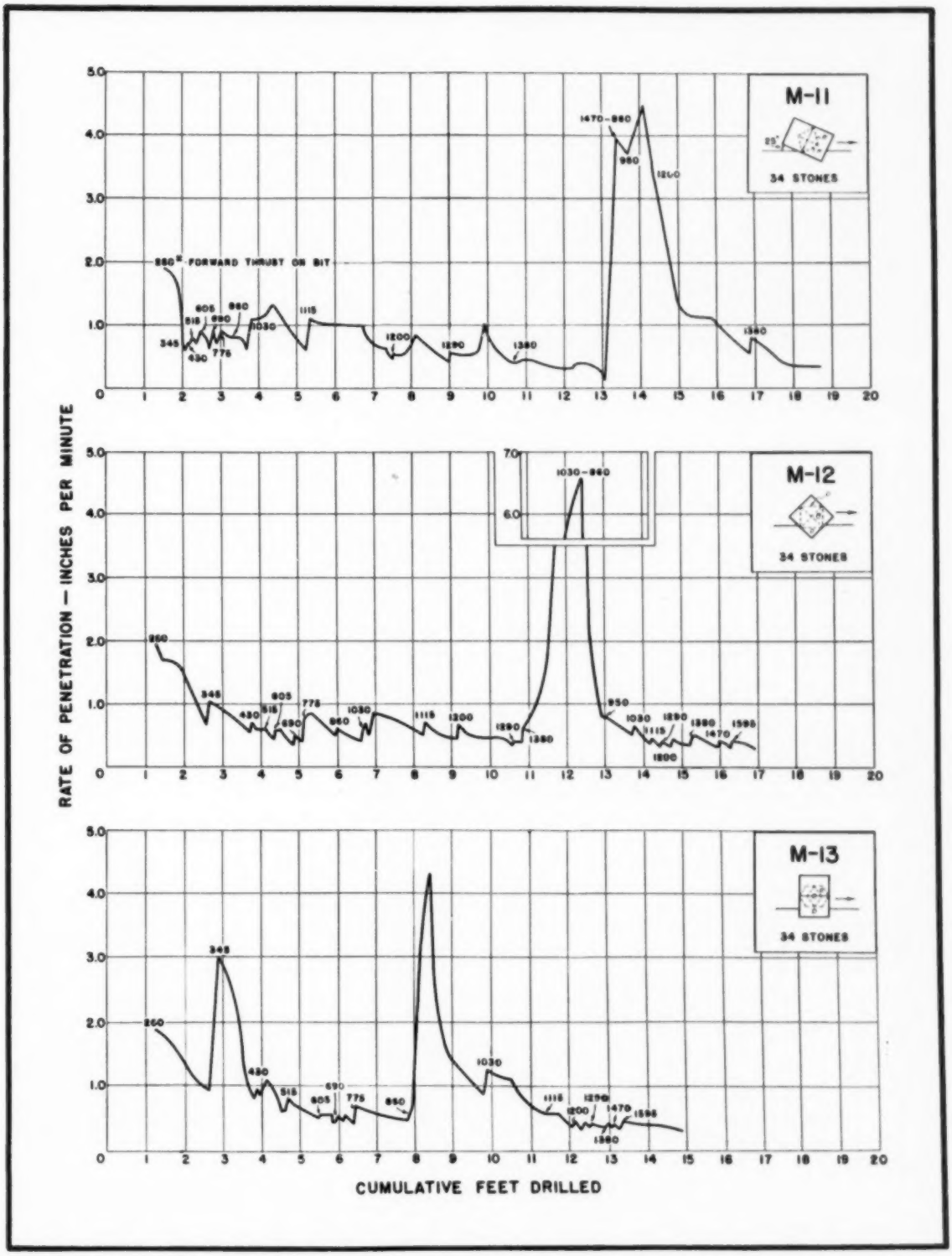

Fig. 4—Test history of EXT diamond drill bits M-11, M-12, and M-13. Constants: 1000 rpm of drill and 2 gpm of water. Variables. Forward thrust of bit and rate of penetration.

Five different orientations were tested:

1—A positive rake off the cube face, with the cube corner or point leading. This gives a negative rake off the octahedral direction, which was found to be a preferable orientation when an octahedral diamond was used. Bits M-11 and M-15, Fig. 1, had this orientation.

2—A negative rake off the cube face with the cube corner or point trailing. This gives a positive rake off the octahedral direction, considered an in-

Fig. 5—Bit M-11 (X4) having point orientation and a positive (gouge) rake, showing formation of abraded flats after 10.9 ft drilling in granite. Rotation counter clockwise.

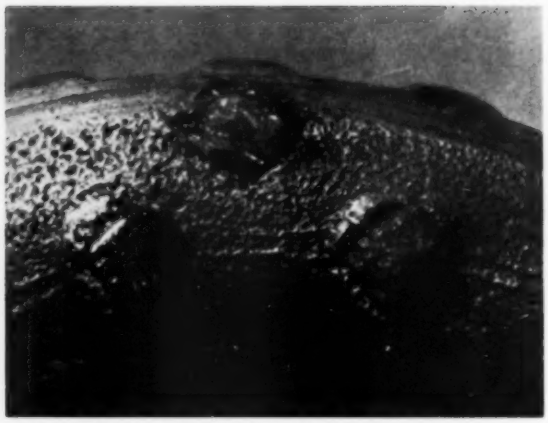

Fig. 6—Bit M-15 (X8) having point orientation and a positive (gouge) rake, showing gage stone fractured along octahedral cleavage after 13.0 ft drilling in granite.

ferior orientation with an octahedron crystal.* See bit M-14, Fig. 2.

* The random orientation by a bit setter of a cube diamond in the round depressions of a standard bit mold would be normally an on point orientation similar to No. 1 and 2.

3—On edge of cube, with edge normal to direction of cutting. Contact then is parallel to the hard direction of a dodecahedral face. An example is bit M-12.

4—On edge of cube, with edge parallel to direction of cutting and hence comparable to drilling along soft direction on a dodecahedral face. An example of this is bit M-13.

5—On edge of cube with a positive *(gouge)* rake off the cube face, equivalent to a negative rake off the hard direction of a dodecahedral face. Examples of this are bit M-16, Fig. 3, and bit M-17.

The only previous known testing of bits having oriented cube diamonds was done in South Africa late in 1948.[7] In conjunction with the Rhokana Copper Corp., the Diamond Research Laboratory tested two EX non-coring bits having cube diamonds, 9 stones per carat, oriented so they would drill on the octahedral plane. These would be on-point orientations. Their bit No. 1, corresponding approximately to bit M-14 described in this paper, drilled a total of 197 ft, whereas bit No. 2, comparable to orientation in bit M-11, Fig. 4, drilled 349 ft. Average footage of a normal production bit of comparable caratage was stated to be only 150. The Rhokana Copper Corp. was enthusiastic about the results and suggested an orientation of the cube diamond on the dodecahedral face, similar to bit M-12, to eliminate the tendency of cube corners to shear off along the octahedral cleavage planes.

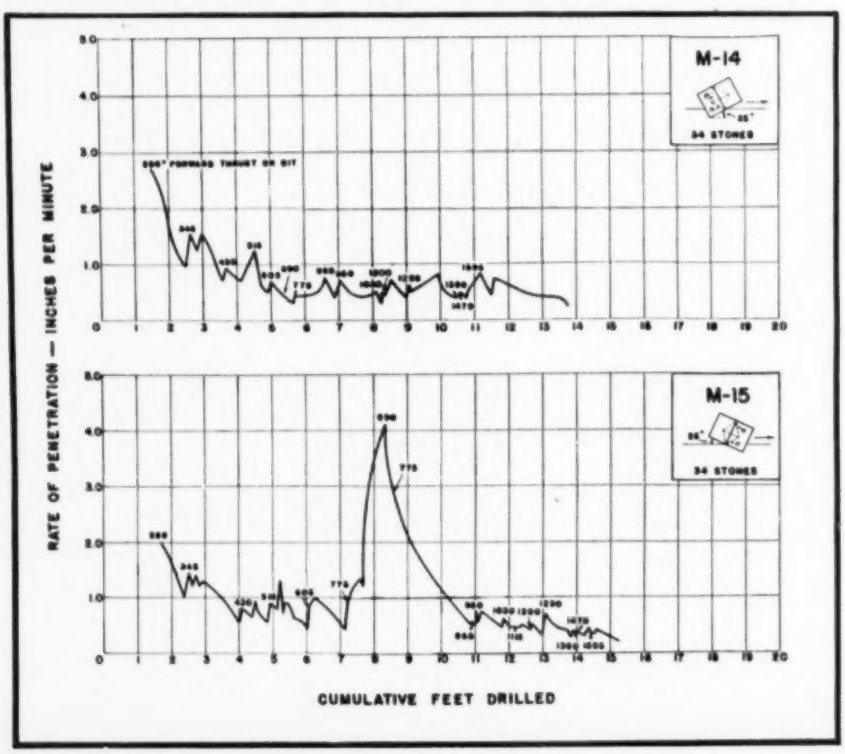

Fig. 7—Test history of EXT diamond drill bits M-14 and M-15. Constants: 1000 rpm of drill and 2 gpm of water. Variables: forward thrust of bit and rate of penetration.

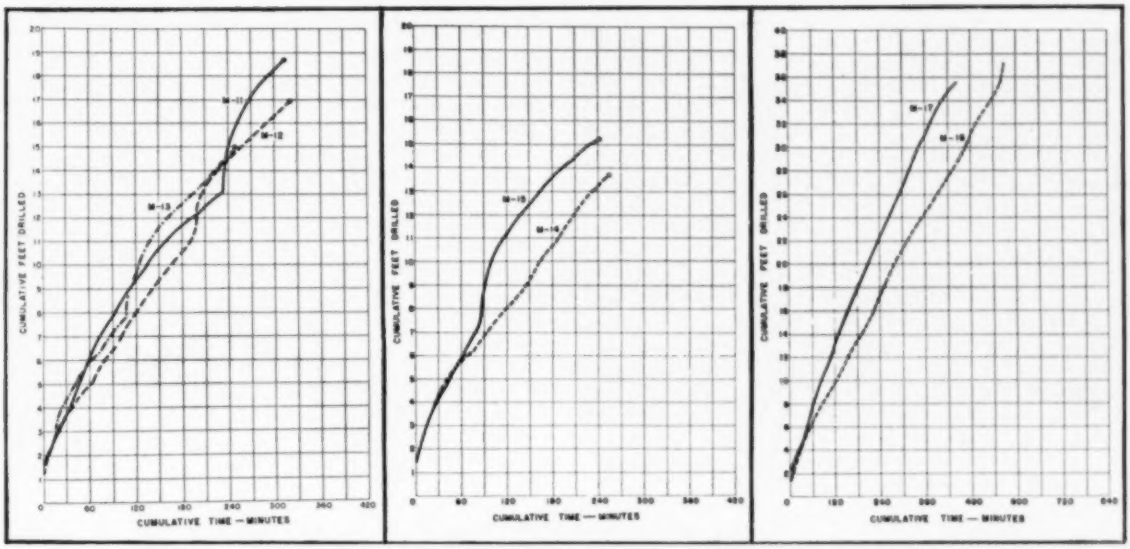

Fig. 8—Cumulative test data, bits M-11 through M-17.

Bits M-11 and M-15 both had point orientation, Fig. 1, with a positive or *gouge* rake off the cube face; hence they should be considered together. This orientation gives a resultant cutting force of diamond on rock that is more or less normal to the octahedral cleavage. Experience in the testing of octahedron diamonds showed this to be a preferential orientation. Such is not the case with cube diamonds. The points wore down rapidly and then tended to form polished flats, Fig. 5, with a resultant decrease in rate of penetration. When a critical pressure was reached, one or more of the diamond points failed, causing destruction of some of the gage diamonds, Fig. 6. This resulted in high, but short-life, advance rates between 13 to 15 ft for M-11, Fig. 4, and 8 to 10 ft for M-15, Fig. 7. Both bits gave approximately the same total footage, 18.7 and 15.3 ft respectively, and the same overall rate of advance, Fig. 8.

Bit M-12 had an on-edge orientation, with the edge normal to the direction of cutting. This orientation should be structurally strong, except for a weakness along the dodecahedral cleavage. In any event, there were indications of diamond breakage early in the life of the bit, and the drilling records showed erratic performance until the breakage became accentuated at about 11.5 ft. Abrasion and polishing of diamonds followed, with pronounced decrease in penetration rate. Overall performance, Figs. 4 and 8, was about the same as for bits M-11 and M-15.

Bit M-13 tested the on-edge orientation with the edge parallel to direction of motion. This is equivalent to drilling along the soft direction of a dodecahedral plane. The diamonds exhibited chipping and breakage in early life, giving periodic high rates of advance. Explanation for this might lie in the fact

Table I. Diamond Wear for Test Bits M-11 through M-17, Set with Oriented Cube Diamonds

EXT Bits	M-11		M-12		M-13		M-14		M-15		M-16		M-17	
Diamond — Type	Congo - Cubes													
Diamond — Size	9 to 10 per carat													
Diamond — Number	34 Stones (26 Crowns; 8 "Kickers")													
Matrix	Powdered Tungsten - Bonding Alloy													
Diamond Orientation and Exposure	+25° off Cube Face; -30° off Oct. Face		45° off Cube Face; On Dodec. Face		On edge of Cube, On Dodec. Face		-25° off Cube Face; +30° off Oct. Face		+25° off Cube Face; -30° off Oct. Face		+22° off Cube Face; -22° off Dodec. Face		Same as M-16	
Ft. drilled	18.68		16.95		14.93		13.72		15.27		37.20		35.62	
Diamond weights	Carats													
Original	3.76	3.76	3.76	3.76	3.48	3.48	3.50	3.50	3.58	3.58	3.50	3.50	3.53	3.53
Final	3.36		3.37		3.16		3.28		3.19		3.15		3.43	
Salvage (reusable)		3.07		2.99		3.07		3.21		2.72		2.95		3.43
Diamond loss	Carats													
Net from wear	0.40		0.39		0.32		0.22		0.39		0.35		0.10	
Total—after salvage		0.69		0.77		0.41		0.29		0.86		0.55		0.10
	Carats per Ft.													
Net from wear	.0213		.0230		.0214		.0161		.0255		.0094		.0028	
Total—after salvage		.0369		.0453		.0274		.0212		.0562		.0148		.0028

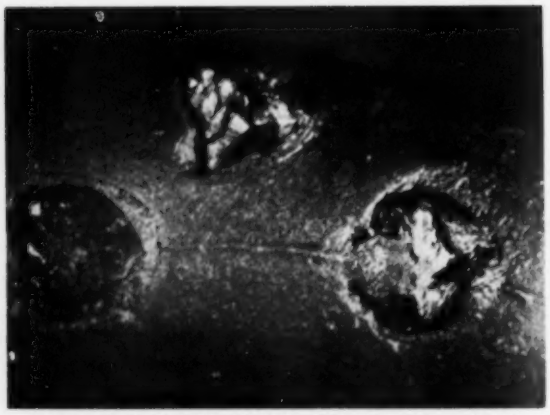

Fig. 9—Bit M-14 (X8) with 30° positive rake off octahedral plane, showing breakage of diamonds along cleavage planes in first foot of drilling during break-in period.

that an edge was sweeping around a circle of small radius, causing torsional forces to act along the edge. Following the formation of sufficient flats by diamond chipping, subsequent abrasion and polishing caused rapid fall off in bit performance. Drilling characteristics and advance were remarkably similar to M-15.

Bit M-14 had diamonds with a point orientation and the cube corner trailing, Fig. 2, and therefore has a positive rake off the octahedral cleavage plane. The corners of the stones started to cleave off in the first foot of break-in drilling, Fig. 9. This confirms previous experience with similar orientation of the octahedron diamonds in bits M-2 and M-4. The tendency for the trailing edge of the stones to cleave away, followed by the formation of polished flats ahead of the broken edge, resulted in a rapid drop in bit performance, Fig. 4, and overall advance, Fig. 8.

Bit M-16, Fig. 4, and M-17 were set with the stones on edge, with the edge itself parallel to the radii drawn from the center of the bit. This is somewhat similar to M-12 with the *important* exception that the diamond was oriented with a 22° negative rake off the dodecahedral plane. This, then, gives a positive, or *gouge*, rake off the cube face, resulting in a strongly backed orientation of fast drilling characteristics. *Thus at one time there is achieved a long cutting edge, a structurally strong orientation, and a gouging or planing action in drilling, all coupled with a diamond wear along the hard direction off the dodecahedral face.* The test results bear out these contentions in a striking manner. Both bits made more than double the advance of the others in the group of seven, totaling 37.2 and 35.6 ft respectively in comparison to a range of 13.7 to 18.7 ft for the other five bits. Drilling performance was remarkably uniform, Fig. 10, except for a high rate of advance near the end of runs of M-16, when a diamond broke and temporarily roughened the others. The typical test history was one of slow gradual abrasion or chipping of the exposed edges, Fig. 11, with very little fracturing or breakage of the individual diamonds, even though the cubes had many points of incipient weakness such as fine hair-like cracks. Eventually the edges either abraded or chipped, Fig. 12, to a degree at which increased sur-

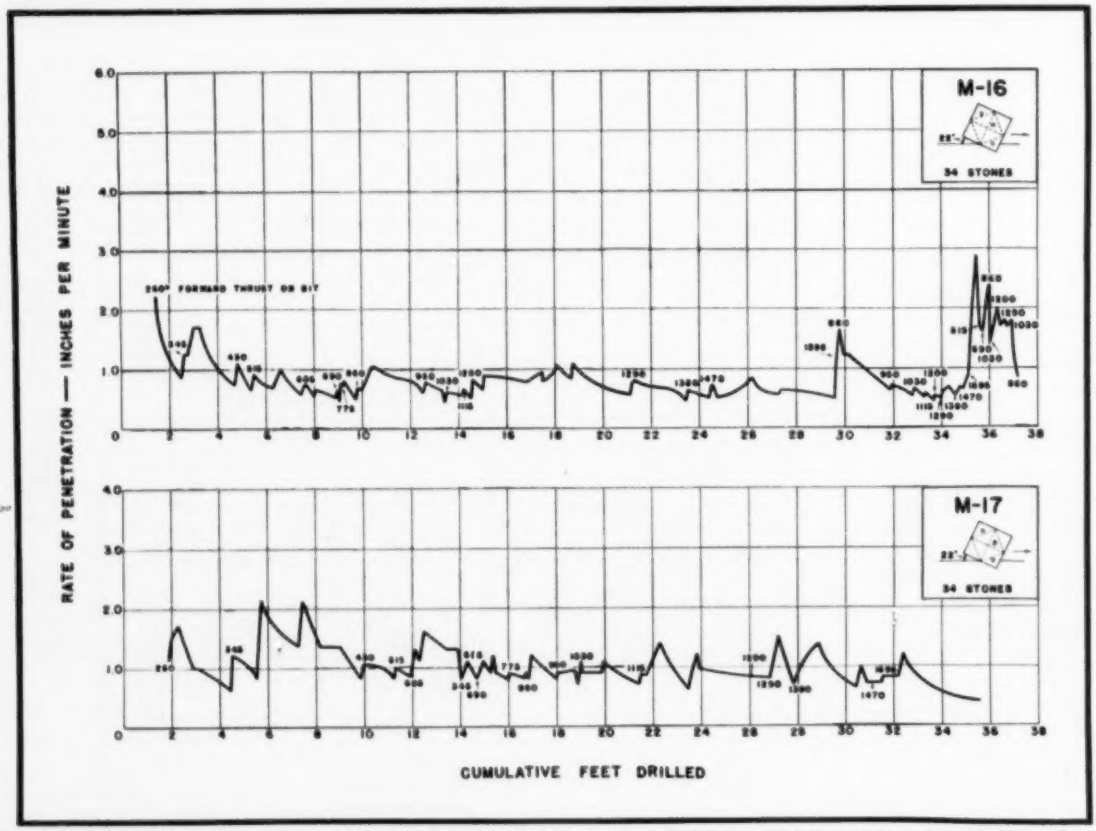

Fig. 10—Test history of EXT diamond drill bits M-16 and M-17. Constants: 1000 rpm of drill. Variables: forward thrust of bit and rate of penetration.

face contact would not permit appreciable penetration of individual diamonds to maintain the set minimum rate of 0.3 to 0.5 in. per min at maximum thrust of 1595 lb. Fig. 8 points up the similarity in performance of the two bits.

Diamond Wear and Loss

The objective of diamond orientation is to increase bit life and drilling rate and at the same time to decrease diamond wear. Often a compromise setting is required to accomplish these objectives, diamond loss being increased to maintain a good drilling advance. This is the case with the octahedral diamond. Such a compromise setting does not appear to be necessary for the cube-shaped stone.

Diamond wear for the group of seven test bits is presented in Table I. Bits M-11 through M-15 have somewhat comparative diamond loss as well as bit performance. Bits M-16 and M-17, with the greatly preferred orientation, show astoundingly low diamond loss. Had drilling stopped with M-16 at 35 ft, before a diamond broke and fractured others, the diamond loss for M-16 would have been approximately the same as for M-17.

All the diamonds from bits M-16 and M-17 were re-usable, and a big advantage of the cube shape is that there are eight similar edges for use in the re-setting process.

Comparison with Octahedron Diamonds

The test work has indicated that for drilling purposes the Congo industrial diamond having a cube form is superior to the Congo octahedron diamond of comparable quality and size. This is true of the drilling rate, total footage, and diamond loss, since the strong structural shape of the cube presents well supported edges, at 90° angles, for cutting. In contrast, the octahedron either has points with acute angles of 70° that are weak and break easily or edges of wide angles, 110°, that do not easily penetrate the rock. Comparative results shown in Table II highlight the superiority of the cubes.

Table II. Comparison of Test Results for Diamond Bits Having Oriented Congo Stones of Cube and Octahedron Shape

Item	Octahedron Diamonds, Bits M-1 Through M-9	Cube Diamonds, Bits M-11 Through M-17
Minimum footage drilled per bit	4.0	13.7
Maximum footage drilled per bit	21.5	37.2
Average footage drilled per bit	12.1	21.8
Average drilling rate, all bits, in. per min	0.5	0.65
Average drilling rate, bits with preferred orientation, in. per min	0.9	1.2
Maximum diamond loss, carats per ft	0.086	0.056
Minimum diamond loss, carats per ft	0.012	0.003
Diamond loss, bits with preferred orientation, carats per ft	0.030	0.009

Fig. 11—Bit M-16 (X6) having edge orientation and positive rake (20° to 25° gouge angle) showing slight wear except for abrasion of inside gage stone after 16.8 ft of drilling in granite.

Fig. 12—Bit M-17 (X4) having edge orientation and positive rake (20° to 25° gouge angle) showing limited abrasion and wear after 20.0 ft of drilling in granite.

Summary

This phase of a research program testing the merits of diamond orientation showed the following results pertaining to the cube-shaped industrial Congo diamond.

1—It is superior to the octahedron, giving a longer bit life at a reduced diamond loss.

2—The cube is a structurally strong stone lending itself to an edge orientation that is fast cutting and of long life.

3—At preferred orientation the edge is set normal to the direction of cutting (parallel to the radii) and has a 20° to 25° positive, or *gouge*, rake off the cube face. This gives a 20° to 25° negative rake off the hard direction of the dodecahedral plane. Diamond loss can be astoundingly low, and the stone presents eight different edges for re-setting.

Acknowledgments

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Fine Coal Drying

by G. A. Vissac

The drying of fine coal involves special techniques, which are discussed and analyzed. Types of dryers employing these techniques are described. Calculations are presented for new methods of dealing with the entrained dust that is always present in fine coal drying operations.

NEW conditions, new requirements, and new methods have increased the demand for more efficient and more economical methods of drying fine coal. Dewatering of larger sizes may reduce the surface moisture to 8 or 9 pct. It is more difficult, however, to dewater sizes below ¼ in., and some filter cakes still contain as much as 20 or 25 pct moisture. Increased freight rates and stricter consumer specifications have resulted in a demand for further reductions in moisture content. This can be obtained only by heat drying.

Most modern methods of heat drying disperse or spread the mass of coal to be dried, in an atmosphere of dry hot gases. The more intimate the contact between coal particles and hot gases, the quicker and more efficient the drying operation will be. Two different techniques are generally employed, using either a fluidized condition or an entrained condition of the coal to be dried.

Fluidized Condition

Fluidization of a body of sand was defined and explained by Fraser and Yancey in a paper published in 1926.[1] This condition was artificially obtained and maintained by proper regulation of the rate of air flowing through the sand body. "The sand bath 'boils' uniformly on the surface," they write, "and feels like a fluid." The fluidization technique was also described and analyzed by Steinmetzer[2] in connection with the operation of an air cleaning table. His main conclusions are as follows: "Fluidity is, for the particles involved, the possibility of motion with minimum friction. . . . Then fluidity requires the introduction of various forms of energy capable of neutralising frictions. Two solutions can be used—air and/or mechanical motions (such as the shaking motion of the carrying deck of the air table). The combination of mechanical and air energy will give the widest margins of size ratios and of bed thickness, translated in capacity per unit area of the carrying table." Richardson and Langston[3] have indicated results obtained with a dryer working with a fluidized bed. They used a vertical tube type of dryer, however, without the assistance of any mechanical energy, and without any lateral motion of the fluidized bed. The capacity of such a dryer is too limited for practical applications, since the speed of the acceptable air currents is held to the speed of fall of the particles involved. Capacities as low as 182 lb of coal per hr per sq ft of dryer area are indicated. As stated by Richardson: "A basic limitation to a fluidised bed dryer is that the velocities of the gas must be held within a definite range; with velocities of 10 ft per second, all coal minus 6 mesh in size will be entrained, and the operation is then similar to that of a Flash dryer."

A fluidized bed must be virtually static. The coal particles simply kept in suspension offer a minimum resistance to the flow of gases, insuring the most favorable conditions for rapid evaporation of surface moisture. However, very wet fine coal, i.e., over 12 pct of surface moisture, will be delivered in the forms of mud balls, or as a soggy, sticky mass, almost impossible to disperse, sticking and acting as a wet blanket on the deck. Strong currents of gases and wide deck perforations will be required to punch holes in the wet mass and gradually loosen and fluidize it. The mechanics of fluidizing a bed of coal in a gas medium for the purpose of obtaining the most efficient drying condition are entirely similar when the fluid used is water and the purpose is to break up and distend a bed of coal to be cleaned so that perfect stratification according to densities will be insured. Purely mechanical energy is used in the basket-type jig, water pulsations in the piston and in the Baum-type jigs. A combination of mechanical motion and of air pulsation offers the most efficient and favorable conditions.

Entrained Condition

The most critical factor to be considered in the design of a dryer employing the entrained condition technique is the speed of the hot gases to be circulated in the drying column. With insufficient gas velocity, excessive amounts of the largest sizes will drop to the bottom of the dryer column without being thoroughly dried. On the other hand, high gas velocity will cause degradation, dust losses, and high power consumption. Figs. 1 and 2, reproduced from Hanot,[4] show the relative importance of speed and temperature for various sizes of particles. It can be seen, for instance, that to maintain in unstable equilibrium particles of ¼-in. size in a gas current at 500°C, a speed of 30 meters per sec, or 6000 fpm, will be required. For ½-in. particles an almost prohibitive speed of 45 meters per sec, or 9000 fpm, will be necessary. In practice, maximum gas velocities of 3000 fpm are recommended; since power increases as the cube of the velocity, it can be seen that beyond certain limits such dryers would not be economical.

If the particles were moving at the same speed as the hot gases they would remain in the same layer of gases, which would quickly become sat-

G. A. VISSAC, Member AIME, is a Consulting Engineer, Vancouver, B.C.

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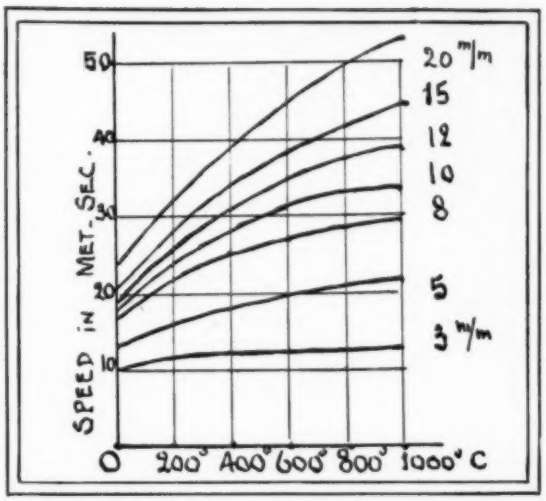

Fig. 1—Speed of gas flow required to float particles of a given size. (After Hanot, *Revue de l'Industrie Minerale*, July 1936.)

urated, limiting the drying possibilities. Accordingly, sliding motion must occur, and this increases with the size of the particle. Fig. 2 gives the relative speeds between gases and particles corresponding to a gas temperature equal to 500°C.

The above discussion clearly illustrates the actions occurring in the mass of coal entrained in a vertical dryer, and the resulting increased friction between particles. This is in fact the exact opposite of the condition in a fluidized bed in which frictions are reduced to a minimum.

Such are the discrepancies in published data dealing with the entrained technique that a simplified theory of the problems involved will be useful.* The weight of a cubical particle of coal of assumed density 1.25 and linear dimensions L, in meters, will be

$$W = 1250 \times L^3 \text{ kgms}$$

The total pressure on such a particle of an air current of speed equal to V will be

$$P = KL^2V^2 \text{ kgms}$$

K is a coefficient which will vary with the shape of the particle and the condition of the air. It expresses the weight of the mass of the atmospheric air column. Under normal conditions this weight is equal to 1.293 kgms and its mass to $1.293/g = \frac{1}{8}$ (approximately), and with various conditions of pressure and temperature $K = \frac{d}{8}$; d = density of actual air or gases and $g = 9.81$ meters (acceleration of gravity). The equilibrium conditions for a particle falling in still air or for a still particle in moving air are

$$P = W \text{ or } 1250\,L^3 = \frac{d}{8} L^2V^2 \text{ or } V = \sqrt{\frac{10{,}000\,L}{d}} \quad [1]$$

This is one of the fundamental equations. For example: $d = 1$, $L = 100$ mesh or 150 microns, and $V = \sqrt{1.5} = 1.24$ meters per sec, equivalent to 4.17 ft per sec.**

For a temperature of 900°F, $d = 0.35$, and $V = 2.1$ meters per sec, or 6.88 ft per sec. For $L = 6$ mesh = 3 mm, the formula gives $V = 18$ ft per sec. According to Hanot, $V = 30$ ft per sec, and according to Richardson, $V = 10$ ft per sec.

Temperature of the Hot Gases

It is easier to dry coal at a high temperature, because the amount of moisture which can be absorbed by unit volume of gases is higher and the time required for a given heat transfer is shorter, but the use of high temperature involves many dangers, namely:

1—Oxidation of the coal, resulting in deterioration of its coking and burning properties.

2—Distillation of some of the hydrocarbons and production of explosive and obnoxious gases, as well as losses in heat values.

3—Losses in overall efficiency in heat balance, due to larger losses in radiation and in entrained heat in discharged or exhausted products.

This can be simply illustrated as follows. In a drying plant treating 200 tons per hr, delivering a dry coal at a temperature of 250°F with an outside temperature of 50°F, the heat losses will amount to 200x2000(250 − 50)0.25 (specific heat of coal) or 20,000,000 Btu's per hr. With a coal at 10,000 Btu's, there would be a total heat efficiency of 50 pct, a loss of 2 tons of fuel per hr. At a value of $4 per ton this is a loss of $8 per hr, $160 per day, and $40,000 per year.

4—Excessive speeds of the hot gases as their density decreases as indicated by the above formula for a given tonnage of coal of given sizes, resulting in greater degradation, higher dust losses, and higher power consumption. In short, everything else being equal, best results and efficiency require use of the lowest possible temperatures.

Vertical Dryers

Early dryers were mostly of the revolving drum type or of the scraper type in which a series of revolving scrapers cause the coal to drop from shelf to shelf against ascending currents of hot gases, Fig. 3. New developments have been using mostly vertical types of dryers in which the wet coal falls or cascades against a current of hot gases or is entrained by it.

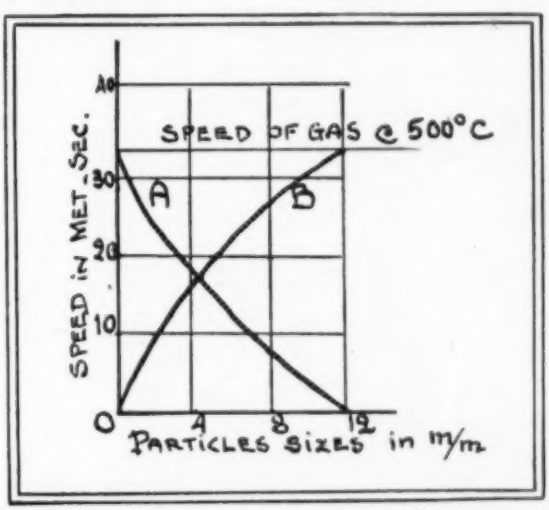

Fig. 2—Relative speeds of gases and particles. (After Hanot, *Revue de l'Industrie Minerale*, July 1936.)

* In these calculations kilograms-meters-seconds are abbreviated as kgms. Speeds obtained in meters per second are also given in their equivalent values in feet per second.

** Richardson[3] gives 4 ft per sec.

In the tower type of dryer illustrated in Fig. 4 the wet coal is fed at the top and cascades by gravity over a series of adjustable inclined plates against a current of hot gases. The larger sizes are extracted by a screw conveyor at the bottom, and the finer particles are entrained and settled in a separator between the dryer and the exhaust fan. In other types of tower dryers star feeders instead of shelves are used, Fig. 5.

In the tube dryer the coal is fed at the bottom of a vertical pipe and entrained with the hot gases. Several types of this system are used in this country. This paper will discuss a well-known German construction, the rapid dryer or Rema-Rosin, see Fig. 6, manufactured by Buttner. The dryer has been described by Hanot.[4]

Hot gases from a pulverized fuel furnace are delivered at the bottom of a tube 33 ft high and 3.3 ft in diam. Wet coal is fed to the bottom of the tube from a surge bin through a screw conveyor and a spreader. Any escaped large piece or ball which might fall is crushed by a wheel located at the lower end of the elbow. Hot gases and entrained coal pass through a separator, the large particles or balls are returned in the feed zone, and the gases and lighter material are delivered to a bin where some of the larger coal will settle. A cyclone collects some of the fine particles entrained, and the balance is settled in a washing tower on the discharge of the exhaust fans. On a bituminous coal containing 20 to 25 pct moisture, tests indicated outputs of from 16.7 to 31.8 tons per hr, with a dry product at 0.82 to 3.04 pct moisture. The fan motor had a rating of 150 kw.

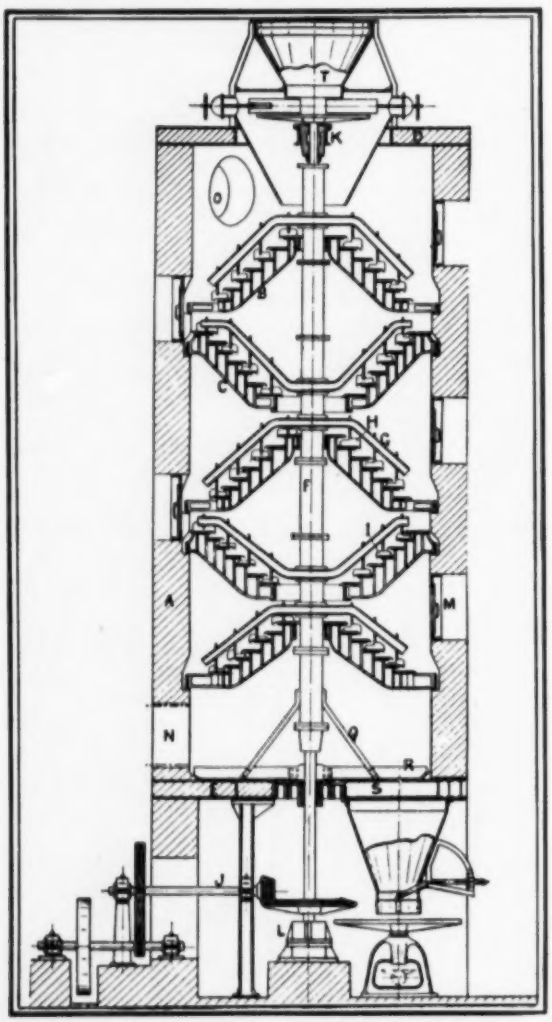

Fig. 3—Vertical dryer. Scraper type. Sahut, Conreur et Cie., France.

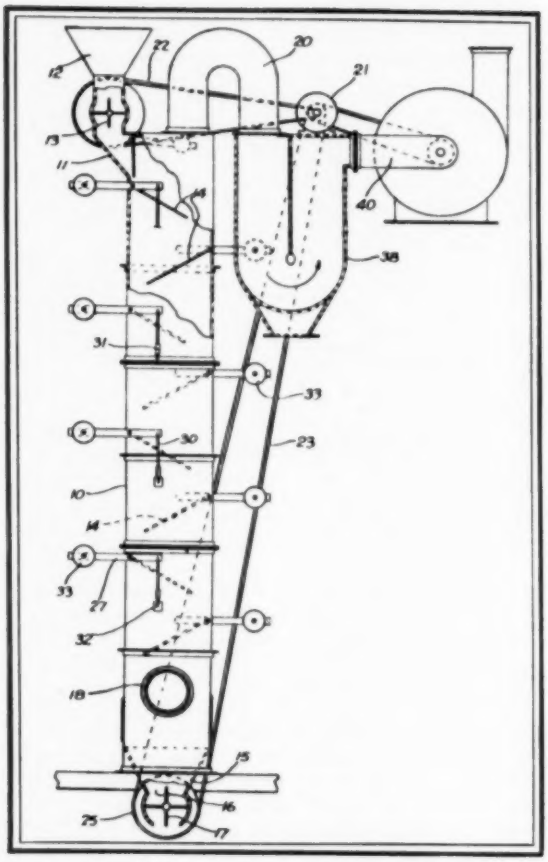

Fig. 4—Vertical dryer. Square tube, gravity type. U.S. Patent 2,317,003.

However, in Europe very few of these dryers are used on bituminous coals. They treat mostly brown coals to remove inherent moisture. Fraser and Driessen[5] report two such large installations for hydrogenating plants at Offleben and Luttzhendorf using seven Rema-Rosin dryers. Each dryer evaporates a ton of water per hour, a low figure compared to the American standard, and also an indication of one of the limitations of the entrained technique. When the drying process is to be followed by a chemical treatment excessive dust must be avoided, as it usually spoils some of the distillation products. In such cases lower gas speeds and therefore lower outputs are required.

The Pulso Dryer

The Pulso dryer, a screen type, is specially adapted to the drying of fine coals. Upward currents of hot gases are used instead of downdrafts used with the screen type for larger coal sizes. The coal is carried over a shaking screen, and fluidization of the bed is obtained by a combination of mechanical motions of the screen and pulsations of the currents of hot gases. Most of the coal is retained over the screen deck, but a variable proportion of the feed drops

through the screen to be dried in the hopper below against an ascending current of hot gases. Because of the perfect dispersion, this new patented method of drying the most difficult sizes under the deck[6] is extremely efficient. Furthermore, this fine coal is trapped under the deck and extracted directly by a screw conveyor, without any exhaust gases, reducing difficulties of dust separation and collection.

Extensive tests and experiments conducted on this dryer at the Ernest mine of the Rochester and Pittsburgh Coal Co. have resulted in the use of a conditioning screen plate, an outstanding improvement. A feed containing a large proportion of fine sticky coals high in surface moisture acts as a wet blanket and is almost impossible to penetrate and fluidize. Stainless steel screen plates with round openings as large as 1 in. at the feeding zone of the dryer have greatly increased its efficiency.

A big advantage of the screen type of dryer with its truly fluidized bed is its capacity to handle a range of sizes up to 4 in. A condition similar to the feldspar bed used with some fine coal jigs is actually realized. The large pieces assist in opening up the bed periodically, and some of the heaviest fine particles, heavier because still overloaded with surface moisture, fall through the bed to be dried against the flow of hot gases in the hopper under the screen. With the entrained condition, as previously mentioned, the practical size limits are around ⅜ in., but with the fluidized condition of the screen-type dryers there is almost no limit to the top sizes which can be treated. All sizes washed, including filter cakes, after whatever dewatering is economically possible, can be fed together to the dryer. Rescreening, if and when required, will be done under better conditions on the dried product, will be more

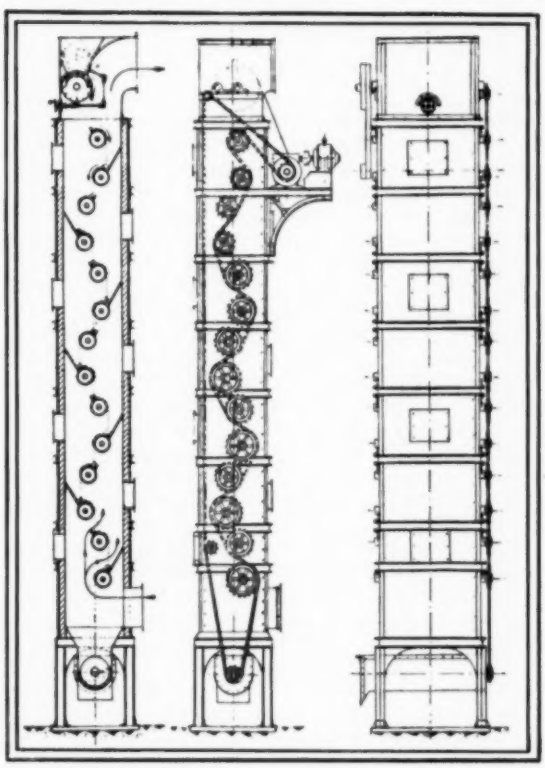

Fig. 5—Vertical dryer. Square tube and star feeders type. Sahut, Conreur et Cie., France.

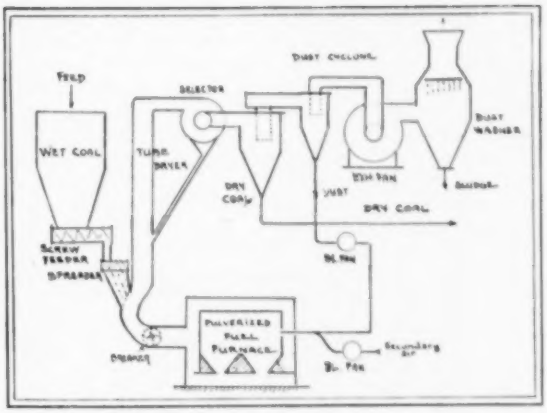

Fig. 6—Vertical dryer. Circular tube type. Rema-Rosin patents by Buttner. (After Hanot, *Revue de l'Industrie Minerale*, July 1936.)

effective, and will result in better appearance for the screen sizes.

Another great advantage offered by such a dryer is the lack of degradation, as indicated in Table I, resulting in fewer fines, less dust, fewer losses in handling and transportation, and, for many users, easier carbonization.

In earlier designs of this type, Fig. 7,[7] with combined use of a blower fan at the furnace and a suction fan at the exhaust, it was found difficult to obtain proper fluidization. Blower fans were then introduced to inject tempering cold air under pressure. With a screening area of 200 sq ft the capacity was only 25 to 30 tons per hr of −¼ in. coals.

Great improvements were obtained by reduction of the screening area to one single deck of 72 sq ft and the use of a larger blower fan between furnace and dryer. But with high drying temperatures this hot gas blower fan was considered a vulnerable feature. It was eliminated by the Rochester and Pittsburgh Co. tests. The new alternative system now

Table I. Rochester and Pittsburgh Coal Co. Ernest Mine Tests on Pulso Dryer

Item	Test G, June 13, 1952		Test L, Aug. 6, 1952	
Feed	62 tons per hr at 10.8 pct moisture		46.2 tons per hr at 13.8 pct moisture	
Product	57 tons pr hr at 2.8 pct moisture		40.0 tons per hr at 2.6 pct moisture	
Evaporation	5.1 tons per hr H_2O		5.2 tons per hr H_2O	
	Screen Analysis of Products			
Sizes	Feed, Pct	Product, Pct	Feed, Pct	Product, Pct
+ ½ in.	2.4	1.2	0.8	0.2
−½ + ¼	43.9	43.2	1.9	1.2
−¼ + ⅛	20.4	21.8	15.2	14.4
−⅛ in. + 10 mesh	14.1	14	29.7	29
−10 + 20	7.0	7	12.8	14.2
−20 + 48	6.2	6.3	20.3	20.5
−48 + 100	2.3	2.6	8.6	9.6
−100 + 200	1.3	1.4	3.9	4.6
−200	2.4	2.5	6.8	6.3

Average results with 10 dryers dealing with various coals and conditions indicate:

Feed	50 to 75 tons per hr
Evaporation	3 to 5.7 tons water per hr
Hot gas temperature	450° to 1150°F
Exhaust temperature	140° to 250°F
Discharged coal temperature	80° to 100°F

consists of one blower fan at the furnace and one exhaust fan at the outlet of the dryer. No moving parts are now left in the path of the hot gases, and temperatures of 1000°F or greater can be used, resulting in a larger output per unit area of the screen dryer up to almost a ton of wet coal per square foot.

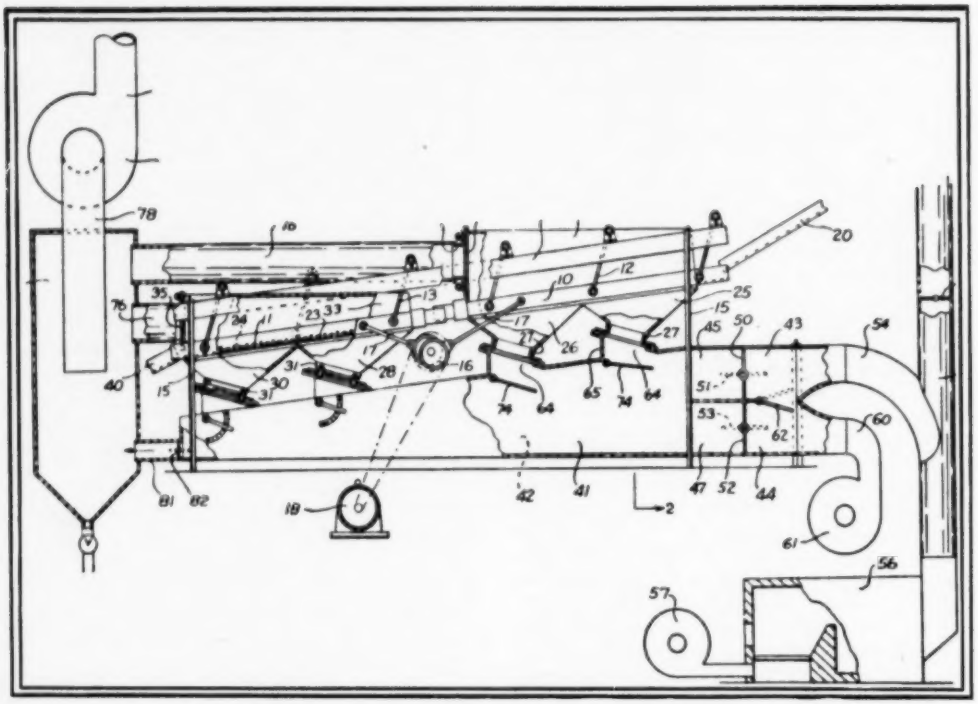

Fig. 7—Screen-type, updraft dryer. U. S. Patent 2,245,881.

In this new system now reduced to two fans, the required fan balance is easier to adjust, insuring minimum pressures at all vulnerable points and eliminating the dangers of fires or explosions, as well as the nuisance and dangers of gas leakages. Typical results of operation are given in Table I. Feeds of over 60 tons per hr of wet coal have been handled successfully, with actual evaporations of from 5 to 5.7 tons of water per hr for a screen area of 72 sq ft. New models with a screening area of 90 sq ft and the same overall dimensions are expected to evaporate 6.75 tons of water per hr. Feeds of fine coals with moisture contents over 14 pct have been handled successfully, without re-circulation.

Table I also indicates the size analysis of the products. Test L deals with very fine coal. Test G deals with a much coarser feed. It can be seen that almost no degradation occurred in the operation, although this is a very friable coal.

A full and detailed description of this type of dryer cannot be presented here. Many mechanical details, such as drives, suspensions, air chests, pulsators, baffles, louvres, ducts, and furnaces, have been gradually improved and developed during the last 20 years and have resulted in a well-tried and efficient drying machine. The new dryer, Fig. 8, uses principles of automatic control originally developed for the downdraft type, namely, automatic control of a variable volume of hot gases at a given and controlled temperature. A new method has also been developed, based on the use of a constant volume of hot gases at a variable temperature, which for some applications results in higher efficiencies and flexibility.

Drying Efficiencies

Although it is difficult to measure all the factors involved in the efficiency of a drying system, elementary laws of fan operation and of thermodynamics can be applied with considerable accuracy. For a given drying problem, at a given temperature, all dryers must use the same weight of hot gases (unsaturated), but the most efficient operation will be accomplished by the system providing the most intimate and the longest contact between the hot gases and the coal particles. Generally speaking, the higher the drying temperature, the lower the heat efficiency, the higher the volume of gases for a given weight, and the lower the fan efficiency. The smaller the section of drying area, the more difficult the dust problems and the higher the resistance. Fan power increases as the square of the resistance.

In pressurized systems recirculation of a fraction of the exhaust gases may be required for safety, but it decreases efficiency. So many Btu's are introduced, but the same amount is withdrawn, and the heat balance is zero. Furthermore, it takes as much power to circulate a given weight of fresh unsaturated gases, but their drying capacity is much higher. D. R. Mitchell shows that "as the percentage of moisture in the drying gases increases, the rate of drying decreases."[3] It may appear that the capacity of a drying unit can be increased simply by the increase of certain factors, or influences, involved, but this may result in nonpractical and inefficient solutions.

The following examples will illustrate methods of calculation used for a drying operation.

In all modern dryers, moisture is removed by absorption from hot gases and evaporation at temperatures under 220°F. Even with hot gases at 1000°F, average temperature of evaporation is 167°F; evaporation at 220°F would require gas temperature of over 5000°F. The fact that coal and exhaust gases are discharged at higher temperatures is only an indication of low efficiency.

A standard unit evaporates 5 tons per hr of moisture; corresponding tonnages of feed may be from

100 tons per hr at 5 pct moisture to 50 tons per hr at 10 pct. Assuming that a dryer treats 50 tons of wet coal per hr and removes 5 tons of moisture per hr, a fan will generally be required, working under 16-in. water gage and delivering 30,000 cfm, all reduced to 70°F and sea level. The corresponding size of motor required is $(30{,}000 \times 16)/4000 = 120$ hp. To double the capacity of the dryer (100 tons per hr feed and 10 tons per hr moisture) will require 60,000 cfm, but since water gage and power vary as the cube of the volume, the size of motor required for double the volume will be $120 \times 2^3 = 960$ hp. In a vertical dryer with a 3-ft diam drying chamber, a volume of 60,000 cfm becomes 180,000 cfm at 1130°F and 120,000 cfm at 600°F. The corresponding air velocities are 25,400 fpm and 17,000 fpm, or for a drying length of 25 ft a drying time of 6/100th to 8/100th sec.

Obviously two 50 ton per hr units will be a better solution than one 100 ton per hr unit.

Dust Collectors

The most difficult problem in operating all fine coal dryers is dust control. As large volumes of gases are used to obtain large drying capacities, it becomes more difficult to collect fine particles of dust produced in the drying of fine coals entrained by the hot gases and to prevent their escape in the atmosphere. The escape of these fine particles is a nuisance and a danger, as well as a loss in saleable coal. A loss of 2 pct in weight, not uncommon with certain types of dryers, on 100 tons per hr, 20 hr per day, 250 days per year, at a value of $4 per ton, amounts to $40,000 per year, or $400,000 over a period of 10 years. Losses of such magnitude should obviously be considered in any competitive analysis of various types of dryers.

Since it is difficult to collect fine coal dust, the amount entrained in the exhaust gases must be held to a minimum. This may be accomplished by trapping of some fine coals below the drying deck, by fluidization of the feed, and by absence of degradation. A certain amount of dust will always be entrained, usually when there are large tonnages and when extreme drying is required. Modern methods of dust control are fully described in a recent publication.[5] In the case of the present dryer, however, for reasons of safety and economy the cyclone is the only practical tool. Simplified calculations are here presented for determining the number and sizes of cyclones required in each specific case.

Assume a particle of coal of cubical shape, of linear dimension L, density 1.25, entering a cyclone of radius R, with a propelled speed equal to V, entrained by a flow of gas traveling at a speed equal to v. The particle will be subjected to two main forces: a force centrifuge F in kilos, and an air pressure P kgs, such as

$$F = \frac{W}{g} \times \frac{V^2}{R} \qquad P = \frac{1}{8} L^2 v^2$$

To achieve separation and settlement of the particle, it is necessary to have

$$F \geqq P$$

or as $W = 1250\ L^3$ and $g = 9.81$, approximately:

$$125L^3 \times \frac{V^2}{R} \geqq \frac{1}{8}\ L^2 V^2 \text{ or } 1000\ \frac{V^2}{R} = v^2$$

A third force, G, or gravity, is practically negligible with the sizes of particles involved; for instance, for $V = 10$ meters per sec, $L =$ microns, and

$$G = \frac{1}{800} P$$

Assume a volume of air of Q cubic meters:

$$v = \frac{Q}{\pi R^2}$$

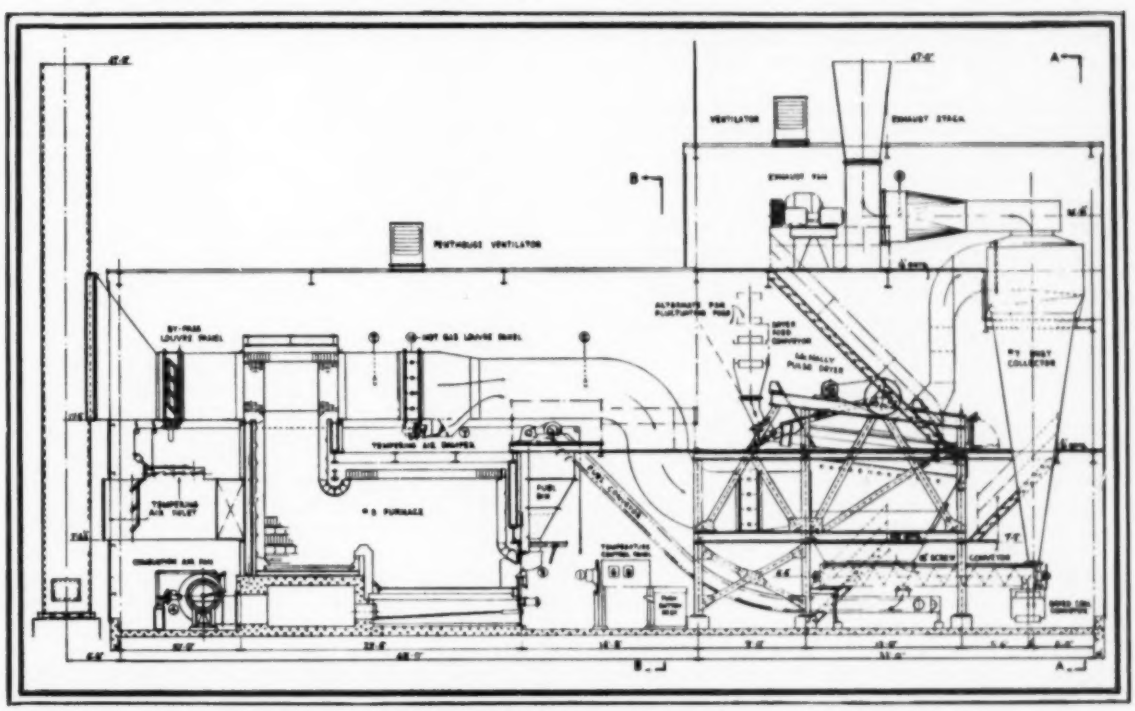

Fig. 8—Typical equipment arrangement of McNally Pulso dryer. Dual fan type. U.S. Patent 2,598,199.

the equation giving the conditions of settlement becomes

$$Q^2 \leqq 10^4 \times V^2 \times R^3 \times L \qquad [2]$$

It is generally assumed that the smaller the diameter the more efficient the cyclone will be. This is a consequence of the equation as given above of the centrifuge force F, in which the radius R of the cyclone enters as a divider. For instance, in the Van Tongeren patents in Europe, tubes as small as 5 in. diam are used, and an installation of dust collecting would contain as many as 680 tubes. As a result, air resistance and power consumption increase out of reason.

It has been found that 7x9-in. diam cyclones are best for the drying problems under consideration. The following examples are given for the above sizes according to the settling formula just presented. V is always taken at 20 meters per sec, or 3600 ft per min, the maximum allowable without undue vibration.

To settle all particles of a size $L = \frac{1}{10^5}$ or 10 microns or larger, in each cyclone it is necessary to use a volume equal to or inferior to Q, such as

$$Q^2 \leqq 10^4 \times L \times V^2 \times R^2$$

$$\text{or with } L = \frac{1}{10^5},\ V = 20,\ R = 1$$

$$Q^2 \leqq 40 \text{ or } Q \leqq 6.32 \text{ mcb per sec, or } 13{,}000 \text{ cfm}$$

Accordingly, if a volume must be cleaned equal to 26,000 cfm, two No. 7 cyclones will be required. On the same basis, if an attempt were made to settle all particles down to 1 micron, the maximum volume allowable per No. 7 cyclone would be

2 mcb per sec or 4200 cfm

and four No. 7 cyclones would be required for a total volume equal to 17,000 cfm. One No. 9 cyclone will efficiently handle almost twice the same volumes, for the same particles, sizes, or the same volumes down to half the size limit the No. 7 can handle.

In the above calculations, a minimum disturbance has been assumed. The required practical conditions are well known: 1—in the leading pipes a maximum speed of 3000 fpm; and 2—in the cyclone a maximum speed of 400 fpm for particles above 10 microns, 200 fpm for particles above 1 micron. Accordingly, the figure obtained from the above formula, on the basis of condition 1 must now be checked for condition 2.

$$V = \frac{Q}{\pi R^2} \text{ with } Q = 6,\ \pi = 3.14\ R = 1$$

$$V = 2 \text{ m per sec} = 396 \text{ fpm}$$

$$Q = 2,\ \pi = 3.14\ R = 1$$

$$V = 0.64 \text{ m per sec} = 130 \text{ fpm}$$

Condition 2 is also well in line.

In the above calculations, it has been assumed that the coal particle travels at the speed obtained at the cyclone entrance. However, in the case of tube dryers in which very high velocities are used, the speed of the particle will not be determined by the speed of the gases at the cyclone entrance, but because of its momentum will race ahead of the air current, just as a bullet races ahead of the propelling gases after leaving the muzzle of the gun. In such cases the momentum of the particle is the dominating factor. It may cause the particle to shoot across the cyclone without a chance of being slowed down and collected.

Actual experience has shown the above formula to be sufficiently accurate to supply easy and convenient guides in the design of the cyclones required in normal coal drying operations. However, it must be pointed out that air currents must be kept within accepted limits to avoid complicated eddies whose motions are impossible to predict.

It is the author's opinion that the Stoke's Law used in some previous calculations, which applies truly to dense media and spherical particles, cannot be used in this case.

Design of Air Cyclones

An air cyclone designed by E. R. McMillan, Manager of the Northwestern Improvement Co. of Seattle, is particularly well shaped and designed for maximum efficiency. Most cyclones use a winding spiral, but the type and modulus of the spiral are very important, as are the length and angle of the bottom cone.

A pulsating flow used in these cyclones is also a new feature of great value,[10] reducing the disturbing action of eddies and accelerating settling actions.

Economics of Fine Coal Drying

Assume that a dryer treats 60 tons per hr of wet coal, reducing the surface moisture content from 9 to 3 pct, or an evaporation of 6 pct or of 3.6 tons of water per hr. (This is in fact a low performance.) The number of Btu's required for the evaporation alone will be $3.6 \times 2000 \times 1000 = 7{,}200{,}000$ Btu's per hr. On the basis of an average overall efficiency of 50 pct, 1000 lb of coal per hr will be required at $4 per ton. This gives the following operating cost:

Wages	$2.00 per hr
Coal	2.00
Power	1.00 (100 kwh at 1¢)
Sundries	1.00
Total	$6.00 per hr at 10¢ per ton of coal treated

The capital cost of such a plant installed will be around $100,000. At 15 pct to cover depreciation and interest charges, there will be an annual cost of $15,000 per year, or on an annual tonnage treated of 300,000 tons (20 hr per day, 250 days) or a capital charge of 5¢ per ton, a total cost of 15¢ per ton. As against the above charges, there will be a saving on freight and handling charges, assuming a total cost for these items of $5.00 per ton, of 6 pct or 30¢ per ton, a credit balance of 15¢ per ton.

But in addition the use of dried coal will result in large savings in installation costs to the consumer. For example, in a coking plant with a unit cost of $120,000 for an oven with a capacity of 18 tons in 18 hr, a reduction of 3 pct in the moisture content may cut the coking time by 50 min, an increased capacity of 5 pct, worth $6000 per oven or $600,000 for a total of 100 ovens (2700 tons per day plant). For a steam power plant with a capacity of 120,000 the capital cost may be around $12,000,000. A moisture reduction of 6 pct will mean an increased capacity of 6 pct, a reduction in capital investment of $720,000.

The figures and opinions presented above, in connection with the influence of heat drying on the coke oven technique, are confirmed and enlarged in

B. Hofmeister's paper,[11] just received, from which the following statements are quoted:

"A low moisture slack will run out faster and loading time can be reduced from 3 minutes to half a minute."

"Coking time can be reduced from 20-22 hours for a slack at 10% moisture, down to 16 hours, for a slack at 4%."

"For a coke plant with a capacity of 1000 tons, a 4% reduction in moisture results in a 25% increased capacity, a saving of .37 mark/ton."

"A decreased consumption of heat results in gas economies of 30 K. Cal/Kg, a saving of .37 mark/ton."

"A longer life for the brick walls results in a further saving of .10 mark/ton."

It is claimed that the greatly reduced loading time, with drier slacks, eliminates danger of explosions.

Desliming Vs Heat Drying

In the case of filter cakes at 20 to 25 pct of surface moisture, increased mechanical dewatering can be obtained by use of additional amounts of clean rinsing water. As much as an additional 2 pct of moisture can be removed in this manner.

Economically speaking, as the slimes removed have an ash content of 25 pct, and as the ash content of the product may have been improved by 1 pct, the corresponding additional loss in the coal treated would be 4 pct in weight. Based on the assumption that coal is $4 per ton, the following balance sheet is presented:

Loss in weight	16¢
Gain in heat value:	
From 1 pct less ash	4¢
From 2 pct less water	8¢
Net loss	4¢

However, on the same basis the balance sheet of a heat drying operation would be:

Gain in heat value from 2 pct less water	8¢
Cost of drying, at 2¢ per ton per 1 pct water	4¢
Evaporation of 6 pct at 10¢ per ton	
Net profit	4¢

This may be the outstanding factor. Increased desliming may not remove more than 2 pct out of the 20 or 25 pct retained by a filter or centrifuge product, and in many cases this may not be enough.

Summary

This paper does not attempt a complete presentation of all types of dryers suitable for efficient drying of fine coals. The writer deals only with dryers he has had an opportunity to study and develop. He has examined the most important problems and presented some simplified methods of calculating the chief factors involved in the hope that it will contribute to further improvements in the techniques of fine coal drying and dust control.

Acknowledgments

The author wishes to express his thanks and appreciation to the staffs of the Riverside Iron and Engineering Works Ltd., Calgary; the Northwestern Improvement Co., Seattle; the Rochester and Pittsburgh Coal Co., Indiana, Pa.; and the McNally-Pittsburg Manufacturing Corp. for their contributions to the successful development of many vital improvements.

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Germanium and Other Elements in Coal And the Possibility of Their Recovery

by A. J. W. Headlee

RECENT interest in germanium centers around its use as a semi-conductor of electricity in electronic devices. It is generally believed that germanium transistors, diodes, triodes, photocells, and rectifiers will largely replace the present-day vacuum tubes. The germanium electronic devices are much smaller, lighter, and more resistant to shock, and they require only a small fraction of current required for conventional devices. Hence only a small amount of heat is given off by the equipment. This development will require many times the quantity of germanium now being produced, which in 1948 was approximately 1000 lb[1] and was obtained almost entirely from zinc ores in Missouri, Kansas, and Oklahoma. No one knows with certainty what the demand will be for germanium by 1956. It may be 40,000 lb.[1] Known concentrations in zinc ores are not sufficient to

A. J. W. HEADLEE is Chief Chemist, West Virginia Geological Survey, Morgantown, W. Va.

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maintain this yearly production of germanium, so that additional sources will have to be developed. Certain coals contain sufficient germanium to warrant their exploitation as a source.

Composition of Coal Ash

Recent researches[3] reported by Headlee and Hunter indicated that several other elements in addition to germanium may be recovered from coal ash. Before the occurrence of germanium in coal is discussed in detail in this paper, the overall composition of the mineral matter in coal will be presented.

Solid columns of coal 3 in. square were sawed from the full height of the seam and brought to the laboratory intact. Cubes of approximately 1 in. were cut at the top, every 3 in. thereafter, and at the bottom. These cubes were tested for porosity, for apparent specific gravity, and for mineral specific gravity. They were then ground to 60 mesh and analyzed for pyritic, sulphate, organic sulphur, and ash content. The ash was then analyzed spectroscopically for 38 elements.

An average analysis of ash for 596 cubes from 35 columns representing 16 coal seams is given in Table I. These coal seams include Redstone (columns 4, 10, 15, and 18), Pittsburgh (columns 5, 7, 8, 9, 11, 16, 17, 21, and 28), Bakerstown (columns 2, 14, and 20), Upper Freeport (column 3), Kittanning (columns 6, 12, and 13), Winifrede (column 27), Cedar Grove (column 34), Campbell Creek (No. 2 Gas) (column 33), Eagle (column 26), Sewell (columns 19 and 23), Beckley (columns 25 and 32), Pocahontas No. 9 (column 30), Pocahontas No. 6 (column 31), and Pocahontas No. 3 (column 29). The map locations are shown in Figs. 1 and 2. The rank of these coals ranges from high volatile A to low volatile bituminous.

Table I. Average Composition of Coal Ash

Oxide	Pct	Oxide	Pct
Lithium oxide	0.08	Chromium trioxide	0.018
Sodium oxide	1.78	Cobalt oxide	0.012
Potassium oxide	1.60	Copper oxide	0.062
Rubidium oxide	0.03	Gallium oxide	0.022
Calcium oxide	2.70	Germanium dioxide	0.013
Strontium oxide	0.41	Lanthanum oxide	0.030
Barium oxide	0.21	Lead oxide	0.048
Magnesium oxide	0.99	Manganese oxide	0.046
Aluminum oxide	30.5	Mercuric oxide	0.013
Silicon dioxide	44.8	Molybdenum oxide	0.016
Iron oxide	14.1	Nickel oxide	0.046
Titanium dioxide	1.51	Phosphorus pentoxide	0.30
Arsenic trioxide	0.018	Silver oxide	0.0015
Antimony trioxide	0.005	Tin oxide	0.022
Beryllium oxide	0.008	Tungsten oxide	0.012
Bismuth oxide	0.002	Vanadium pentoxide	0.050
Boric oxide	0.097	Zinc oxide	0.051
		Zirconium dioxide	0.029

Cadmium, cerium, and tellurium were looked for in each sample but were below the limits of detection, at 0.005, 0.01 and 0.05 pct respectively.

Sodium, potassium, rubidium, calcium, magnesium, silicon, chromium, and manganese are present in coal ash to a lesser extent than in the earth's crust. Aluminum, iron, titanium, beryllium, cobalt, copper, molybdenum, nickel, phosphorus, vanadium, and tungsten are 1 to 10 times as rich in coal ash as in the earth's crust, while lithium, strontium, barium, silver, arsenic, bismuth, boron, gallium, germanium, mercury, lanthanum, lead, tin, zinc, and zirconium are 10 to 185 times as concentrated in coal ash as in the earth's crust. Several of these 15 elements should be commercially recoverable from coal ash.

The above statements are based on an average for 35 columns. Some individual columns have values 2 to 5 times greater than the average so that ash in a certain area is richer in a particular element than the ash in another area.

Germanium in Coal

Germanium concentrations[4] occur in the coal in a manner entirely different from that of any other element. These concentrations are shown in Table II.

Table II. Germanium Dioxide Content of Redstone Seam, Monongalia County

Depth, In.	Ash, Pct	Germanium Dioxide Content	
		Ash, Pct	PPM of "Ash-Free" Coal
1	14.7	0.017	29
3	13.1	*	
6	24.4	*	
9	5.5	*	
12	4.8	*	
15	6.0	*	
18	7.3	*	
21	79.1	*	
24	54.1	*	
27	5.3	*	
30	5.3	*	
33	4.7	*	
36	23.1	*	
39	4.8	0.014	7
42	5.8	*	
45	14.8	*	
48	7.3	*	
51	13.7	0.021	33
54	28.0	0.033	141

* Below the limits of detection, 0.005 pct.

The germanium is concentrated in the top and bottom of the coal seam. In most cases the top and bottom inch of coal showed the highest concentration of germanium. The adjacent sample in each case is nearly 3 in. away. To find the vertical variation in greater detail, the top 3 in. of column No. 19, Sewell seam, and the bottom 4 in. of column

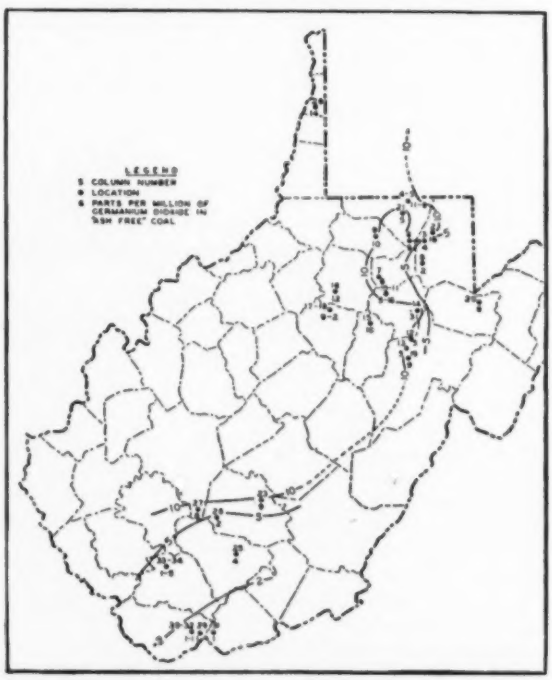

Fig. 1—Map showing isometric germanium dioxide lines for coal seams, full seam averages.

No. 2, Bakerstown seam, were sampled every ¼ in. and analyzed for germanium. These two sections were selected because of their high germanium content. The data are recorded in Tables III and IV.

The germanium content of the Sewell seam when calculated as percent ash is the highest in the fourth to eighth ¼-in. section and decreases from that high point. The germanium content when calculated as parts per million of ash-free coal is the highest in the upper ¼-in. of the top of the Sewell seam, while the third ¼-in is the highest in the bottom of the Bakerstown, with decreasing values above and below this. It would appear that the germanium was adsorbed by the coal as water entered it carrying germanium salts in solution. The other possibility is that the coal adsorbed germanium from gases containing volatile germanium compounds.

Screen and Float-Sink Tests on Bottom 3 In. of Bakerstown Seam

Twenty-four hundred pounds of the bottom 3 in. of the Bakerstown seam were obtained from a mine near Reedsville, Preston County, W. Va. The coal was screened and the various sizes were sampled and tested for ash and germanium dioxide content, see first part of Table V.

Table III. Germanium Content of Twelve ¼-In. Sections From the Top of Column 19, Sewell Seam

		GeO_2	
Section, ¼-In.	Ash, Pct	Ash, Pct	PPM in Coal (Ash-Free)
First from top	8.49	0.13	121
Second	5.94	0.18	107
Third	7.08	0.11	84
Fourth	1.89	0.27	46
Fifth	2.20	0.24	54
Sixth	2.13	0.20	44
Seventh	2.25	0.22	51
Eighth	2.10	0.14	30
Ninth	1.89	0.08	15
Tenth	1.65	0.024	4
Eleventh	1.74	0.010	2
Twelfth	1.62	0.012	2

Table IV. Germanium Content of Fifteen ¼-In. Sections From the Bottom of Column No. 2, Bakerstown Seam

		GeO_2	
Section, ¼-In.	Ash, Pct	Ash, Pct	PPM in Coal (Ash-Free)
Fifteenth	2.41	0.022	5
Fourteenth	4.75	0.008	4
Thirteenth	3.47	0.037	13
Twelfth	7.83	0.025	21
Eleventh	5.71	0.045	27
Tenth	4.00	0.08	33
Ninth	4.15	0.09	39
Eighth	2.79	0.25	72
Seventh	4.09	0.20	85
Sixth	4.78	0.19	96
Fifth	6.58	0.12	87
Fourth	8.99	0.10	99
Third	6.15	0.104	127
Second	6.15	0.083	54
First from floor	5.59	0.064	38

The screened fractions were washed with heavy media (magnetite) in a laboratory size washer. The float fractions were sampled and tested for ash and germanium dioxide content, second part of Table V. The shale refuse from the washer did not contain detectable quantities of germanium. It is to be noted that the germanium content of the ash is inversely proportional to the percent of ash in the coal, and the germanium content when calculated on the "ash-free" coal basis remains nearly constant. It appears that the germanium is in the organic matter. The removal of as much of the ash as possible without losing coal prior to burning will concentrate the germanium in a much smaller quantity of the ash. Screening, then, does not separate the coal into high and low germanium coals according to particle size. However, those screenings which have the lowest ash content will have the highest germanium concentration in the ash.

Table V. Germanium Dioxide Content of Screened and Washed Fractions of Bakerstown Coal Sample

Fraction		Bottom 3 In. Germanium Dioxide		
Screen Size, In.	Gravity	Ash in Coal, Pct	Ash, Pct	PPM of "Ash-Free" Coal, Pct
+2		27.2	0.028	103
+1-2		40.4	0.015	102
+½-1		25.8	0.034	118
+¼-½		19.7	0.050	123
+⅛-¼		18.3	0.050	112
+1/16-⅛		13.0	0.076	114
−1/16		16.8	0.054	109
(+2)	−1.34	9.3	0.10	103
(+1-2)	−1.30	10.0	0.10	111
(+½-1)	−1.34	8.5	0.10	93
(+¼-½)	−1.30	8.6	0.11	104
(+⅛-¼)	−1.29	6.8	0.13	95
(+⅛-¼)	−1.42	10.2	0.10	114
(+1/16-⅛)	−1.35	9.6	0.096	102
(+2)	−1.21	5.4	0.15	86
(+2)	−1.29	5.9	0.17	107
(+1-2)	−1.29	6.3	0.18	121

The concentration of germanium by screening depends on the particle size distribution of the shale. But screening is not a satisfactory way of separating mineral matter from coal. Washing technique does present an efficient method of removing mineral matter from coal, thus permitting the germanium to remain with a minimum amount of ash after coal combustion. Further static gravity separations are being made of the last 10 samples in Table V using potassium carbonate solutions of 1.28, 1.29, and 1.34.

Areal Variation in Germanium Content

The areal changes in germanium content of the coal are similar to those for volatile matter. These variations are shown in Fig. 1 for 35 samples from 16 seams.

Individual seams show a more consistent variation as shown for the Pittsburgh coals in Fig. 2. The germanium content of the coal appears to be proportional to the volatile matter. There is some indication that coals of lower rank than high volatile A bituminous coal decrease in germanium content.

Germanium in Ashes From Combustion Equipment

In the combustion of the coal varying amounts of the germanium are retained by the ash and may be concentrated under certain conditions. Extensive investigations were made in England[4] in the past few years to determine whether or not these concentrations were sufficient to warrant recovery. Certain flue dusts in producer gas sets were found to be rich enough in germanium, about 0.5 pct, for its economic recovery. This dust collects in the colder portions of the flues conducting the gases away from the set if they are allowed to cool. If the hot gases are burned before cooling, germanium collects in the colder portions of the furnace flues. The total recovery from this source amounts to relatively small quantities of germanium.

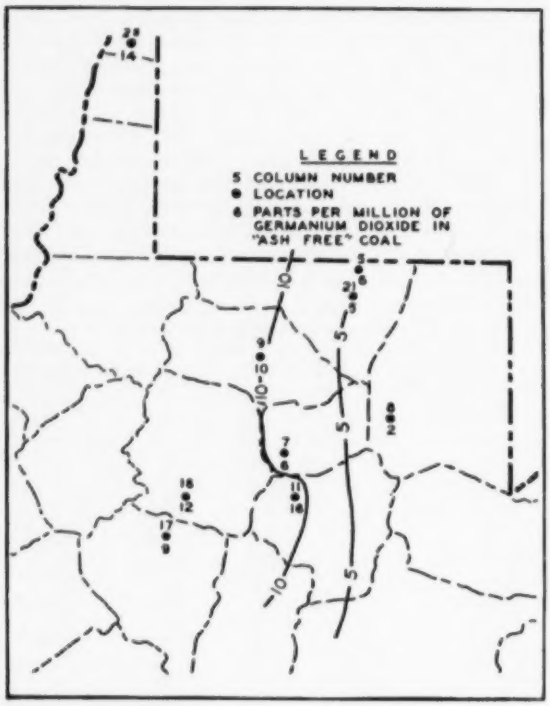

Fig. 2—The above map indicates the germanium dioxide in the Pittsburgh seam.

It does not appear likely that extensive recovery will be obtained except by the selection of low-ash highly germaniferous coals for specific combustion equipment in which a minimum of germanium is lost through the stack. Even in carefully controlled laboratory ashing operations some of the germanium is lost when the temperature never exceeds 740°C. It is not known with certainty just how much of the germanium is lost in combustion equipment. Estimates vary from a few to 70 pct of the germanium. However, when germanium recovery from coal ash becomes a reality, then compounds can be added to the coal to retain most of the germanium in the ash if necessary.

Methods of Recovery

Recovery of germanium from coal ash is long and tedious. This cost is reflected in the value of the finished product. Preparation of the germanium dioxide is further complicated by the fact that it must be of extremely high purity before it is suitable for use in electronic devices. A high percentage of the value of the product is required to offset the cost of purification. If it were necessary for common salt (sodium chloride) to be that pure it would probably cost a similar price per ton. Arsenic is particularly troublesome and difficult to remove. The final traces are removed by distillation of germanium tetrachloride through copper turnings. English[1,4] methods of recovery from coal ash have been reported. American[1] methods have not been reported in detail. Methods of recovery will probably include the following steps: 1—fusion or its equivalent, 2—precipitation of the germanium as the sulphide, 3—conversion of sulphide to anhydrous chloride, 4—high efficiency distillation of chloride, 5—high efficiency distillation over copper turnings, 6—hydrolysis of chloride to germanium dioxide.

The end use of the germanium is as a metal. The dioxide is easily reduced to the metal with hydrogen above 700°C. Most of the electronic manufacturers prefer to produce their own metal from the oxide. Certain other metals have to be added in very small quantities to produce germanium metal with the desired electrical characteristics.

Economics

Germanium dioxide sells for $142 per lb, or $284,000 per ton. The price schedule for coal ash will probably be based on the germanium content of the ash. One prospective buyer has suggested that ash containing 0.2 pct germanium is worth $75 per ton. Considerable tonnage of ash could now be prepared whose value might be $25 or more per ton. Probably the minimum germanium concentration in coal ash having a recovery value is 0.05 pct. This lower limit will change as recovery research progresses and may go considerably lower. It is too early in this development to establish a firm price or lower limit.

Tests to date indicate that germanium will seldom be present in coal in sufficient quantities to mine and process coal for germanium recovery alone. Most of the mining cost will still have to be recovered by burning coal as a source of heat in conventional combustion equipment and recovering the germanium from the ash.

Conclusions

Germanium is commercially recoverable from coal ash. The bottom 3 in. of the coal seam can be selectively mined, washed, and burned to produce an ash rich in germanium. Certain coal utilization equipment concentrates germanium so that recovery is feasible from this source. Processes are available for recovery of germanium from the ash.[4,6]

The recovery of germanium will be the incentive for the recovery of other elements from coal ash.

Acknowledgment

This paper is a review of the literature. The only unpublished data are those in Table V, regarding which the author wishes to express his appreciation to G. R. Spindler and H. W. Ahrenholtz, Jr., West Virginia University, School of Mines, for their aid and the use of their heavy media washer. Mr. Leslie Larsen, President, Fairfax Mining Co., very generously supplied the coal sample. The author gratefully acknowledges the aid of the staff of the West Virginia Geological Survey, and particularly that of Paul H. Price, for permission to publish this paper.

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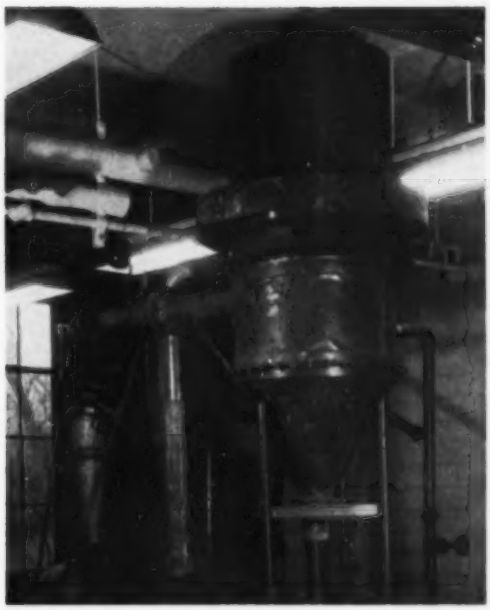
The Application Of Centrifugal Forces To Gravitational Classifiers

by Robert C. Emmett

and

Donald A. Dahlstrom



FOR many years gravitational classification has been employed as a basic tool in beneficiation of minerals and coal. While improvements have been made to increase efficiency and fields of application, the basic flow pattern has remained relatively unchanged in most gravitational classifiers. Feed slurry is introduced into a settling pool wherein the coarser solids settle against either an upward or horizontal fluid current, or a combination of both, and are withdrawn in a concentrated underflow stream. The finer solids below the classification point possess too low a settling velocity and thus are primarily swept out through a weir overflow by the bulk of the fluid. Because of its simplicity, efficiency, and low power and maintenance costs, gravitational classification finds a wide acceptance for separations in the range of 20 to 325 mesh.

Two factors resulting from the basic flow pattern employed in gravitational classification present difficult problems for certain installations. If classification at 60 to 200 mesh of separation is desired, capacity in terms of gallons of overflow per minute per square foot of settling pool area must be low to allow required solids to settle to the underflow. If a thousand or more gal per min of feed slurry must be handled, either a very large classifier or a number of smaller units must be employed, taking up floor space and head room. Examples of this are the large classifiers, wetting tanks, or settling tanks used in the coal industry for recovery of process water from fine solids. Units as large as 50 or 60 ft in diam and 60 ft high, requiring substantial amounts of steel, are not uncommon.

A second factor is the lack of *self-regulation* in gravitational classification. As overflow rate increases, mesh of separation increases correspondingly. This normally heightens difficulties of beneficiation. If a fine mesh of separation is required at varying feed rates, the problem is more than doubled in complexity.

One of the primary advantages of gravitational classification is low power requirement. Gravity feeds or very low pressure heads are employed. However, fluid energy is still present in the inlet stream and provision must be made to dissipate it properly to obtain uniform upward or horizontal currents. Rather than waste this energy, it would be desirable to utilize it to secure a greater degree of self-regulation and simultaneously increase classification capacity per square foot of area. A free vortex could be realized by a tangential entry of feed, resulting in generation of centrifugal force to complement force of gravity in its action. With greater force it should be possible to gain finer separation at any overflow rate per square foot of settling pool area as compared with conventional gravitational classification. Furthermore, as feed rate increases, centrifugal force will also increase and at least partially offset faster currents sweeping solid particles to the overflow. A more stable classification point should result.

The liquid-solid cyclone is the most common application of a free vortex in the beneficiation field.[1] Many previous papers have emphasized uniformity of flow pattern within the cyclone which is largely responsible for the sharp classification that can be obtained.[2-4] If a free vortex can be maintained in a non-pressurized vessel with a free surface it may be possible to realize a third advantage of sharper classification. This open-top cyclone would then achieve a minimum energy loss comparable to that with conventional classifiers, and a mesh of separation coarser than 150 mesh, extending the range of application of the free vortex. This is not presently possible with ordinary cyclones.

To test the applicability of the free vortex to gravitational classification, a 30-in. diameter open-top cyclone was constructed at Northwestern University. The cylindrical section was 30 in. high and the conical section included an angle of 60°. The feed nozzle was a standard 4-in. pipe entering tangentially, with centerline 17½ in. below the top of the cyclone. The conical section was flanged near

R. C. EMMETT is Development Engineer, the Eimco Corp., Chicago, and D. A. DAHLSTROM, Member AIME, is Associate Professor, Chemical Engineering Department and Director of Research and Development, the Eimco Corp., Chicago.

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Table I. Results for Open-Top Cyclone at Varying Feed Rates and Volume Splits, All Solids 2.83 Sp Gr Dolomite

	Run 113	Run 114	Run 115	Run 116	Run 117	Run 118	*Run 119	Run 120
Slurry Rate, GPM								
Feed	138.2	142.9	157.6	85.5	209.3	159.0	162.3	165.7
Overflow	117.0	121.0	132.0	71.0	183.2	129.1	148.8	127.2
Underflow	21.2	21.9	25.6	14.5	26.1	29.9	13.5	38.5
Solids Concentration, Wt, Pct								
Feed	22.6	23.4	24.5	23.5	22.6	22.0	19.10	22.0
Overflow	9.06	9.16	9.31	9.2	9.00	9.60	9.74	9.03
Underflow	66.1	69.0	69.4	66.0	74.5	57.6	75.7	52.0
Solids Rate, Tons per Hr								
Feed	9.06	9.80	11.42	5.91	13.87	10.12	8.81	10.56
Overflow	2.79	2.92	3.26	1.73	4.37	3.28	3.85	3.04
Underflow	6.29	6.88	8.16	4.18	9.50	6.84	4.96	7.52
Recovery in Underflow, Wt, Pct								
+20 mesh, U.S. standard	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
20x40	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
40x60	99.7	99.8	99.7	99.9	99.7	99.7	99.8	99.9
60x100	96.6	96.2	96.2	99.3	93.7	95.7	93.2	95.8
100x140	80.4	80.3	81.7	87.8	68.3	74.2	53.0	77.1
140x200	45.4	52.6	56.2	61.1	42.8	46.0	19.9	45.5
−200	6.0	9.3	9.9	12.4	7.7	12.6	4.3	17.0
Diameter of Separation, Microns	138	131	142	111	149	143	141	145
Equilibrium Particle, Microns	99	94	92	88	108	104	129	104
Classification Efficiency at 100 Mesh, Pct	83.1	81.0	84.6	76.9	84.1	82.9	91.5	78.9
Liquid Level above Bottom of Horizontal Overflow Section, In.	11	12	17	4.5	28	17	17	

* Overloaded underflow discharge.

the apex to permit insertion of different 60° underflow cones with outlet diameters ranging from ⅝ to 1¾ in.

Four-inch copper tubing was used for the overflow, the inlet being located at cyclone centerline on a plane level with the bottom of the feed nozzle. A 90° elbow joined the vertical overflow section to a horizontal outlet tube passing through the cylindrical section wall. The centerline of the horizontal section was 8½ in. above that of the feed nozzle. Withdrawing the overflow above the feed nozzle preserves the potential head of the fluid, making it possible to utilize gravity flow in plant applications.

The horizontal overflow section, because of its location cross-current to the flow, caused considerable turbulence with resulting breakdown of flow pattern along the downstream side of the pipe. Classification was impaired and capacity per unit of fluid energy was lowered. An airfoil of galvanized iron was constructed following dimensions given for a standard streamline form with a fineness ratio (length to maximum diameter) of 2.[5] The airfoil enveloped the horizontal section from junction at cyclone wall to edge of elbow. Turbulence along the trailing edge was greatly diminished, although further improvement is desirable. Data will be shown to indicate the advantages obtained by this design improvement.

The cyclone was originally constructed with a peripheral weir discharge 7 in. above the top of the overflow horizontal section. This discharge was used at high feed rates, but it was found that it contained coarser solids than the regular overflow stream. Accordingly, a 2-ft extension was added to the cylindrical section, making a weir discharge unnecessary. Evidence of the superiority of this construction will also be proved later by the experimental data. The frontispiece illustrates the open-top cyclone construction after the 2-ft cylindrical extension had been added.

All runs were made on slurries containing ground agricultural dolomite of 2.83 sp gr. All particles were uniform in shape and specific gravity, and less than 1 pct were coarser than 20 mesh. The solids were charged to a 250-gal open tank equipped with a high speed mixer and water added to obtain correct solid concentration. A slurry pump fed the pulp to the open-top cyclone through a hand-operated throttling gate valve for control of flow rate. The overflow, underflow, and weir discharge returned to the slurry tank through launders or sheet metal conduits. Before any analyses or rate samples were taken, the system was allowed to pump for several minutes to attain equilibrium.

Samples were taken from all discharge streams to secure data on size distributions and solid concentrations. Rates were determined by collecting a discharge stream in a weighing tank for a timed interval. Solid concentration and size distribution of the feed stream were obtained by back calculation from the discharge streams.

Effect of Overflow Rate and Volume Split: After many exploratory and test runs, it was established that the sharpest and finest classifications were obtained with a streamlined horizontal overflow section, and only one overflow discharge originating at the cyclone centerline. Several runs, therefore, were made at varying feed rates and volume splits to the underflow with constant feed solid concentration. Results are indicated in Table I. Range and average size distribution for feeds of Table I are given in Table II and indicate sufficient uniformity. It will also be observed in Table I that feed solid concentration varied between 22.0 and 24.5 pct only for all runs except run 119, which exhibited an overloaded underflow discharge and will be considered separately.

Table II. Size Distribution of Feed Solids in Table I

Size Fraction	Range, Wt, Pct	Average Wt, Pct
+20 mesh U.S. standard	0.5 to 0.8	0.6
20x40	30.8 to 38.0	34.4
40x60	18.1 to 20.8	19.4
60x100	6.5 to 10.2	8.3
100x140	2.9 to 3.7	3.3
140x200	2.6 to 3.6	2.9
−200	29.1 to 33.8	31.1

Two particle dimensions were calculated from the data: 1—diameter of separation, the diameter in microns at which 1½ pct by weight of the overflow solids would be coarser; and 2—equilibrium particle, the diameter of the particle in microns which is in equilibrium with the fluid currents at the critical separation point within the open-top cyclone. Theoretically, it is the particle size whose settling velocity is exactly equal to and opposite that of the radial velocity of the fluid at the radius

separating the inner and outer spirals of the free vortex within the cyclone under the conditions of operation. It can be calculated as follows. Any water withdrawn in the underflow must carry with it *at least* the same percentage of feed solids (heavier than the slurry density) of each diameter as the per cent of feed water reporting to the underflow. Therefore, the recovery per cent in the underflow for the equilibrium particle will be:

$$X = \frac{100-Y}{2} + Y = 50 + \frac{Y}{2} \qquad [1]$$

Where X = weight per cent recovery of the equilibrium particle size in the underflow

Y = per cent of feedwater reporting to the underflow.

Accordingly, from the underflow recovery curve for each run, the equilibrium particle size would be determined at the above per cent.

The diameter of separation and equilibrium particle size for Table I are plotted against the overflow rate in gallons per minute in Fig. 1. A small increase in equilibrium particle size is observed with an increase in flow rate, while diameter of separation shows a slightly steeper correlation. Both curves appear to be asymptotic in nature; the reverse is true for the conventional liquid-solid cyclone. Both curves would exhibit a decrease as overflow rate increased for a pressurized cyclone of fixed dimensions, in as much as centrifugal force increases faster than radial forces tending to sweep particles to the overflow.[3, 8] Because of the relatively small fluid forces involved in the open-top cyclone, probably a relatively high percentage of fluid energy is consumed in overcoming wall friction, which is negligible in the conventional cyclone. Thus a smaller percentage of fluid energy is converted into centrifugal force so that the increased radial forces at higher flow rates cannot be entirely offset. However, it may be possible by improved design, i.e., smaller included angles and better streamlining of overflow section, to reduce

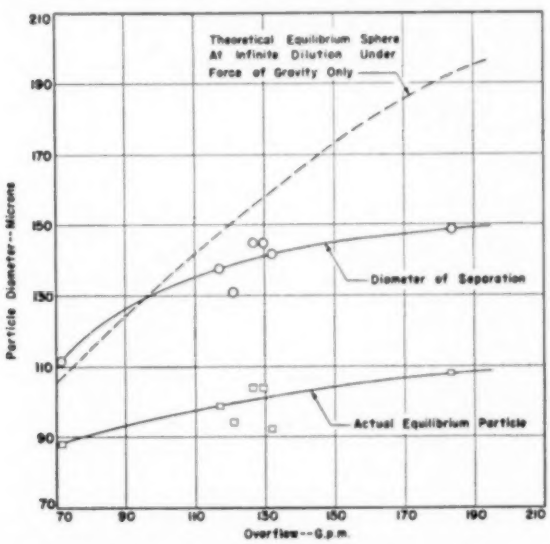

Fig. 1—Diameter of separation and equilibrium particle size as a function of overflow rate for the 30-in. experimental open-top cyclone. Feed solid concentration range, 22.0 to 24.5 wt pct.

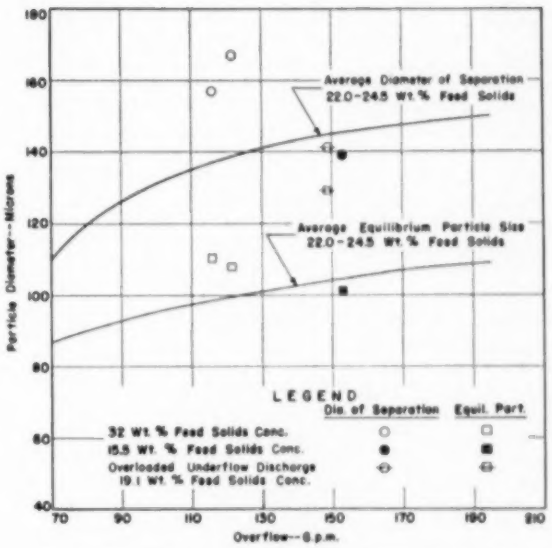

Fig. 2—Effect of feed solids concentration and type of underflow discharge on diameter of separation and equilibrium particle size for the 30-in. experimental open-top cyclone.

wall friction and obtain a diameter of separation or equilibrium size independent of overflow rate.

The theoretical equilibrium sphere curve at infinite dilution under the force of gravity only for the conventional gravitational classifier has been included in Fig. 1. This was done by calculating the diameter of a sphere of specific gravity 2.83 whose settling velocity at infinite dilution is equal to the overflow rate in cubic feet per minute divided by the cross-sectional area in square feet of the open-top cyclone. This curve would represent theoretical minimum equilibrium particle size for gravitational classifiers operating on this solid and would naturally be exceeded under hindered settling conditions.

It should be pointed out that the equilibrium particle for conventional gravitational classifiers is generally felt to be best expressed by diameter of separation. All particles coarser than this size will be given a greater settling velocity, by the constant force of gravity, than that of the opposing fluid currents. However, this is not true for a free vortex, as neither the centrifugal force nor radial velocity is constant across any normal plane to the centerline. Accordingly, equilibrium separation occurs at the division between the inner spiral, directed toward the overflow, and the outer spiral, directed toward the underflow. It is hypothesized that this is best represented for a free vortex by the equilibrium particle size calculated from Eq. 1. On the basis of equilibrium, comparison would have to be made between diameter of separation for gravitational classifiers and what has been called equilibrium particle size for the open-top cyclone. However, owing to the general recognition of the importance of diameter of separation in analyzing classification results, comparison can better be made on this basis.

It will be observed that the equilibrium curve at infinite dilution for gravitational classifiers for the most part lies well above the diameter of separation of the open-top cyclone. In a gravitational classifier there would definitely have been hindered settling with feed and overflow solid concentrations

similar to those found with the open-top cyclone. Thus improvement will be even better. Furthermore, the slopes of the two curves of Fig. 1 for the open-top cyclone are considerably less than the theoretical curve for infinite dilution under the force of gravity only. It can be safely concluded that a free vortex pattern has been established which effectively adds centrifugal force to force of gravity to obtain a finer classification point.

It will be noticed in Table I that underflow solid concentration was allowed to vary from 52.0 to 74.5 pct without any appreciable effect on its diameter of separation or equilibrium particle size. This is similar to results obtained with conventional liquid cyclones.[8]

Sharpness of classification is indicated by efficiency of separation calculated from a formula by Taggart.[9] Efficiencies ranged from 76.9 to 84.6 pct, which is generally considered high compared with efficiencies for gravitational classifiers. However, classification is greatly affected by size distribution and feed solid concentration, so that as Taggart cautions,[9] efficiency of classification expressed as a percentage is relatively unimportant by itself. Other data are necessary for purposes of comparison.

As a better indication of the sharpness of separation with the open-top cyclone, it will be observed in Table I that the diameter of separation is equal to or finer than 100 mesh in all cases. Simultaneously, it was found that the percentage of the −200 mesh fraction of feed recovered in the underflow was in all cases equal to or very slightly greater than the per cent of feed water recovered in the underflow. As this represents the absolute minimum recovery of any size fraction, it is apparent that the underflow recovery curve breaks from a maximum of 100 pct to an absolute minimum over a very close size range.

Effect of Feed Solids Concentration and Type of Underflow Discharge: Three runs, reported in Table III, were made at feed solids concentrations ranging from 15.5 to 32.1 pct. Diameter of separation and equilibrium particle size were calculated as indicated in Table III and are also plotted in Fig. 2 as a function of overflow rate. The average curves of Fig. 1 for diameter of separation and equilibrium particle size at 22.0 to 24.5 pct feed solids concentration are reproduced on Fig. 2 for comparison.

From Fig. 2 it is apparent that feed solids concentration of 32 pct increased diameter of separation and equilibrium particle size over those at 22.0 to 24.5 pct as would be expected. Undoubtedly this is due to more intense hindered settling conditions. Diameter of separation increased by an average of 17.4 pct, while equilibrium particle size increased only 9.8 pct. These increases are well within values that would be predicted by hindered settling theory on gravitational classifiers.

With the above results, it would be logical to expect a lowered classification efficiency. As seen from Table III, efficiency was only 75.6 pct as compared with an average of 82 to 83 pct for the 22.0 to 24.5 pct feed solid concentration. Naturally, some of this drop in efficiency is also due to removal of a greater percentage of feed water in the underflow with the heavier solid concentration.

The results of Run 123 at only 15.5 pct feed solids concentration are slightly lower than the average conditions of 22.0 to 24.5 pct feed solids concentration. However, as only one run was made, this could be a statistical deviation. From previous conventional cyclone work, increase in equilibrium particle size on this material would be expected to begin at about 20 pct feed solids concentration.

During the exploratory runs overloaded underflow discharges were obtained in a few cases, but it was observed that overflow solids did not appear to increase in top size. This is contrary to experience with conventional cyclones with overloaded discharges wherein solid recovery in the underflow is very severely decreased and most of the particles are lost to the overflow. Run 119 of Table I was performed with an overloaded underflow discharge at 19.1 pct feed solids concentration to secure quantitative information. It will be observed from Fig. 2 that equilibrium particle size increased about 25 pct over that of the average results at 22.0 to 24.5 pct feed solids concentration and a vortex underflow. However, diameter of separation was slightly below the average, indicating that coarser particles were not being lost to the overflow. From these results it is also apparent that classification occurred over a relatively narrow size range, with the result that classification efficiency at 100 mesh was a very high 91.5 pct.

The explanation for the continued high recovery above the vortex underflow mesh of separation, even with overloaded discharges, probably lies in the much lower centrifugal forces utilized in the open-top cyclone. Thus when an overloaded underflow occurs, centrifugal force is not sufficient to *twist-off* the inner spiral at the conical section apex to severely impair solids discharge. A similar phenomenon has been experienced with large conventional cyclones, 36-in. diam, operating at only 3 lb per sq in. gage feed pressure.

It is not argued that an overloaded discharge will never increase the mesh of separation of an open-top cyclone. Naturally it would be possible to feed solids at such a rate that they could not be withdrawn fast enough through an undersized underflow. Rather, it indicates that the open-top cyclone should be able to handle a wider variation in solids rate than the conventional cyclone without impairing mesh of separation. Furthermore, it was observed that there was never plugging of the underflow spigot, as can occur with some gravitational classifiers.

Effects of Non-Streamlined Overflows: Several runs were made initially without the streamlined horizontal overflow section. Data for Runs 104 and 106 are given in Table IV with operating conditions comparable to those of Table I and Fig. 1 using the streamlined overflow. Diameter of separation averaged 17 pct higher and equilibrium particle size 9.5 pct higher than the corresponding values for

Table III. Open-Top Cyclone, Effect of Feed Solids Concentration on Classification, All Solids 2.83 Sp Gr Dolomite

	Run 121	Run 122	Run 123
Slurry Rate, GPM			
Feed	159.1	157.6	164.0
Overflow	121.2	116.0	153.1
Underflow	37.9	41.6	10.9
Solids Concentration			
Feed	31.35	32.1	15.5
Overflow	10.98	11.72	8.95
Underflow	69.6	67.2	69.6
Solids Rate, Tons Per Hr			
Feed	15.50	16.19	7.11
Overflow	3.55	3.74	3.66
Underflow	11.95	12.45	3.45
Diameter of Separation, Microns	167	157	139
Equilibrium Particle, Microns	108	110	102
Classification Efficiency, Pct at 100 Mesh	75.8	75.4	86.4
Liquid Level Above Bottom of Horizontal Overflow Section, In.	17	14	21½

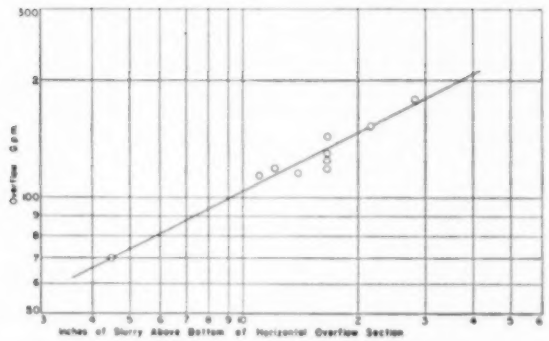

Fig. 3—Energy requirements for open-top cyclone, experimental 30-in. unit.

the streamlined overflow. As would be anticipated, therefore, classification efficiency was also lowered.

The flow pattern was obviously more turbulent with the non-streamlined overflow, indicating that more of the initial fluid energy was not available for the centrifugal force. Accordingly the above results are to be expected. Consequently every effort should be made to present as little obstruction to the free vortex flow pattern with the open-top cyclone. This necessitates not only streamlined overflows but also smooth wall construction and continuous taper of the conical section.

Effect of Weir Overflow Discharges: Several runs were also made initially with a peripheral overflow weir discharge as well as a simultaneous central non-streamlined overflow discharge. Table V shows results for Runs 107 and 112, whose operating conditions are similar to runs of Table I and Fig. 1 with only a central streamlined overflow discharge. Equilibrium particle size averaged 16.2 pct higher than with the streamlined central overflow. However, diameter of separation was considerably above that of the streamlined central overflow. Weir discharge averaged 99.7 pct higher, the simultaneous central overflow averaged 43.4 pct higher, and the composite averaged 57.8 pct higher. While the non-streamlined overflow contributed to a portion of this injury, it is obvious from the previous section that greatest impairment arose from the weir discharge.

The peripheral weir receives fluid from the outer radius of the free vortex. This radius possesses the lowest centrifugal force of the outer spiral wherein all separation must be achieved. Consequently, the higher diameter of separation is not surprising. However, the free vortex pattern is also distorted by such a weir discharge. A high percentage of the energy of the fluid so discharged is not available for the creation of centrifugal force within the unit. Consequently, the central overflow discharge also suffers severely. This is easily proved by the higher diameter of separation in the central overflow of Runs 107 and 112 when compared with results of Fig. 1. Accordingly finest separations can be expected in the open-top cyclone when the optimum creation of a free vortex is followed. A single central overflow from the inner spiral of the vortex is therefore desirable.

Energy Requirements: Because of the very low feed pressures involved, below 1.5 lb per sq in., it was impossible to obtain reliable data on feed pressures to the open-top cyclone. Height of the outer liquid surface, which is proportional to energy

Table IV. Open-Top Cyclone, Sample Results without Streamlined Overflow Section and Comparison with Streamlined Overflow Results

	Run 104	Run 106
Slurry Rate, GPM		
Feed	116.7	125.2
Overflow	94.9	102.8
Underflow	21.8	22.4
Solid Concentration, Wt, Pct		
Feed	17.5	20.7
Overflow	6.83	7.01
Underflow	50.2	60.7
Solids Rate, Tons Per Hr		
Feed	5.73	7.47
Overflow	1.68	1.87
Underflow	4.05	5.60
Diameter of Separation, Microns	149	146
Diameter of Separation with Streamline Overflow, Microns	129	132
Equilibrium Particle Size, Microns	110	98
Equilibrium Particle Size with Streamline Overflow, Microns	94	96
Classification Efficiency at 100 Mesh, Pct	71.0	68.3
Estimated Classification Efficiency at 100 Mesh with Streamlined Overflow and Similar Underflow Solid Concentration, Pct	76.0	80.0

requirements, was accordingly noted with the single streamlined overflow runs, see Tables I and III.

The best correlation for energy requirement was obtained by plotting the log of overflow rate in gallons per minute against the log of the height of slurry above the bottom of the horizontal overflow section as indicated in Fig. 4. A line of slope 0.5, which would be anticipated by fluid mechanics theory, fit the data with sufficient accuracy. The equation for this straight line is:

$$\text{gpm} = 115\sqrt{F} \qquad [2]$$

where gpm = overflow rate, gal per min
F = height of slurry above bottom of horizontal overflow section, in feet of overflow slurry.

In a previous paper, the expression $\text{gpm}/\sqrt{F}$ has been termed the capacity ratio. For a conventional cyclone, it was found equal to the following:

$$\frac{\text{gpm}}{\sqrt{F}} = K\ (eb)^{2.0} \qquad [3]$$

where e = overflow diam, in.
b = inlet diam, in.
K = capacity ratio proportionality constant.

Table V. Open-Top Cyclone, Sample Results with Weir Discharge and without Streamlined Overflow Section. Comparison with Streamlined Overflow and without Weir Discharge at Same Capacity

	Run 107	Run 112
Slurry Rate, GPM		
Feed	200.0	212
Overflow	133.3	128
Weir	47.0	63.3
Underflow	19.7	20.7
Solid Concentration, Wt, Pct		
Feed	16.05	18.3
Overflow	7.71	9.9
Weir	6.93	9.2
Underflow	64.5	67.1
Solid Rate, Tons Per Hr		
Feed	8.93	11.09
Overflow	2.70	3.38
Weir	0.85	1.54
Underflow	5.38	6.17
Diameter of Separation, Microns		
Overflow	197	232
Weir	295	302
Composite	222	250
Without weir discharge and with streamlined overflow at same capacity	149	150
Equilibrium Particle Size, Microns	118	133
Equilibrium Particle Size, Without Weir Discharge and with Streamlined Overflow at Same Capacity, Microns	108	109
Classification Efficiency at 100 Mesh, Pct	80.5	81.6
Estimated Classification Efficiency at 100 Mesh without Weir Discharge and with Streamlined Overflow at Same Capacity, Pct	84.0	84.0

Using this equation with appropriate diameters of feed and overflow for the open-top cyclone, the capacity ratio proportionality constant would be 9.38. Conventional cyclones will normally exhibit values of this constant from 5.5 to 6.5. Accordingly, on the basis of the above equation the experimental open-top cyclone permitted a 44 to 71 pct higher capacity per unit of energy input. However, it is felt that the last equation will not hold for the open-top cyclone. Instead, either the 0.9 exponent will be changed to a higher value or the capacity ratio will be a function only of the overflow diameter. Further experimental work will be needed to verify this. At any rate, the higher capacity ratio is to be expected from previous comments. Wall friction percentage is felt to be considerably more important with the open-top cyclone than with conventional cyclones. As it is easier to overcome wall friction than to generate a free vortex, the higher throughput per unit of input energy for the open-top cyclone is entirely logical.

It should be pointed out that the greatest submergence used in these runs was 28 in. above the bottom of the horizontal overflow section. To obtain feed pressure at the feed entrance to the open-top cyclone, it is necessary to add the distance from the centerline of the inlet to the bottom of the horizontal section of the overflow. The experimental unit had a separation of only 6½ in., making the maximum feed pressure less than 3 ft of overflow slurry. This compares favorably with any gravitational classifier. Furthermore, as the overflow is discharged near the top of the cyclone, much of the original feed energy is still available in the form of potential head.

Comparison with Other Gravitational Classifiers: Data from the section of classification in the *Handbook of Mineral Dressing* by Taggart was used to compare the results of the open-top cyclone with other gravitational classifiers.[7] Summaries of performances of four different classifiers will be found in Table VI together with representative runs on the open-top cyclone. Diameter of separation, classification efficiency, overflow gallons per minute per square foot of settling pool area, and tons of feed solids per hour per square foot of settling pool area were also calculated and listed in Table VI. It should be emphasized that performance runs were selected so that operating conditions were as closely comparable as possible with respect to solid specific gravity, feed solids concentration, and diameter of separation. Because of this specification, it was not possible to obtain comparisons for all types of gravitational classifiers.

It is not intended to imply that the open-top cyclone is entirely superior to other gravitational classifiers. Naturally each application will designate the type of classifier to be used. Rather, the comparisons will be made with respect to what is believed to be the specific advantage of the open-top cyclone: higher capacity per square foot at a relatively stable mesh of separation, with possibly sharper classifications.

All performance tests but one exhibited a mesh of separation of 100 or 150 mesh, comparable to results with the open-top cyclone. Rake classifiers averaged 2.9 gpm of overflow per sq ft of settling pool area with a high of 4.6. Bowl-rake classifiers averaged 4.2 gpm per sq ft with a high of 7.35. The one test on the Allen cone gave 1.8 gpm per sq ft while the Callow tank yielded 3.1 gpm per sq ft. The open-top cyclone ranged from 14.5 to 37.3 gpm per sq ft with an average of 27.4. This average is 552 pct higher than the highest average of the Bowl-Rake classifiers. As has already been pointed out, moreover, the open-top cyclone can accept a severe increase in feed rate with only a small change in diameter of separation. Thus the 37.3 gpm per sq ft is a realistic figure which is 789 pct above the highest gravitational classifier average. If higher slurry levels above the overflow pipe can be tolerated, still higher capacities per square foot can be obtained.

Rake classifiers averaged 0.47 tons of feed solids per hr per sq ft of settling pool area with a high of 0.55, bowl rake classifiers 0.29 tons with a high of 0.55, the Allen cone 0.28 tons, and the Callow tank 0.18 tons. The open-top cyclone averaged 2.11 tons per hr per sq ft for the 5 representative runs, with a high of 3.30. This average is 349 pct above the highest exhibited by rake classifiers. Because of the classification stability of the open-top cyclone with

Table. VI. Comparison of Operating Data for Industrial Classifiers with Open-Top Cyclones

Type of Solid	Solid Sp Gr	Solids, Wt, Pct Feed	Solids, Wt, Pct Over-flow	Solids, Wt, Pct Under-flow	GPM Feed	GPM Over-flow	GPM Under-flow	Solids, Tons Per Hr Feed	Solids, Tons Per Hr Over-flow	Solids, Tons Per Hr Under-flow	Diameter of Separation, Microns	Classification Efficiency, Pct*	Pool Area, Sq Ft	Over-flow, GPM Per Sq Ft	Solids, Tons Feed Per Sq Ft Per Hr
Rake Classifier—Taggart, Table 6, Section 8															
Siliceous	2.7	27.6	20	82	433	402	31	36.3	23.0	13.3	149		87.3	4.6	0.42
Quartz		37.2	23	74	338	272	66	41.2	18.3	22.9	173	40.5(100)	87.3	3.1	0.47
Siliceous	2.6	45.8	20	73	241	146	95	38.3	8.3	30.0	237	46.5(65)	91.2	1.6	0.42
Quartz	2.7	46.7	12	82	290	200	91	50.2	6.5	43.7	142	47.7(100)	91.2	2.2	0.55
Bowl-Rake Classifier—Taggart, Table 18, Section 8															
Siliceous		18	10	74	828	780	48	37.5	20.8	16.7	149	67.2(100)	106	7.35	0.35
Siliceous	2.6		10			641		63.8	17.1	66.7	144	38.0(100)	153	4.2	0.55
Siliceous	2.7	21	5	70	665	645	20	14.6	8.35	6.25	102	70.9(150)	491	1.3	0.03
Siliceous		17	9	72	856	789	67	41.7	18.8	22.9	146	71.3(100)	201	3.9	0.21
Allen Cone—Taggart, Table 26, Section 8															
		35	14	70.5	70.5	51.8	18.7	7.91	2.00	5.91	151	40.9(100)	28.3	1.8	0.28
Callow Tank—Taggart, Table 30, Section 8															
		18.4	13.7	54	169	156	12.9	8.87	5.83	3.04	108	78.3(150)	50.03	3.1	0.18
Open-Top Cyclone															
Dolomite	2.83	22.6	9.00	74.5	209.3	183.2	26.1	13.87	4.37	9.50	149	84.1(100)	4.91	37.3	2.82
Dolomite	2.83	23.5	9.20	66.0	85.5	71.0	14.5	5.91	1.73	4.18	112	82.9(140)	4.91	14.5	1.20
Dolomite**	2.83	19.1	9.74	75.7	162.3	148.8	13.5	8.81	3.85	4.96	138	91.5(100)	4.91	30.2	1.80
Dolomite	2.83	32.1	11.72	67.2	157.6	116.0	41.6	16.19	3.74	12.45	154	75.4(100)	4.91	23.6	3.30
Dolomite	2.83	15.5	8.95	69.6	164.0	153.1	10.9	7.11	3.66	3.45	135	86.4(100)	4.91	31.2	1.45

* Numbers in parentheses indicate the mesh at which efficiency is calculated.
** Run operating with an overloaded underflow discharge.

respect to variations of feed rate, higher solids rates than the above average are entirely feasible.

It is believed that open-top cyclone capacity per square foot of area can be increased still further. Pressurized cyclone performance has indicated that inlet and overflow nozzles should cover approximately 60 pct of the cyclone diameter to obtain minimum classification points. In other words,

$$\frac{e + 2b}{D} = 0.6$$

where D = cyclone diameter.

In the experimental open-top cyclone with 4-in. inlet and overflow nozzles, only 40 pct of the 30-in. diam was consumed. Accordingly, it is felt that a 20-in. diam could have been employed in this work. This would decrease cross-sectional area to only 44.5 pct of the experimental unit while increasing overflow and solids rates per square foot to 2.25 times those actually experienced. Overflow rate of 84 gpm per sq ft may be entirely realistic. This is 1900 pct above the highest average of the gravitational classifiers.

With the previously mentioned warning that efficiency of classification can be misleading, it will be noticed that only one gravitational classifier test exhibited a percentage as high as the open-top cyclone. Classification efficiencies at mesh of separation for gravitational units ranged from 38.0 to 78.3 pct, while the open-top cyclone ranged from 75.4 to 91.5 pct. From this comparison a sharper classification with the open-top cyclone appears possible.

No comparison was made with respect to power requirements, but it is believed that the maximum submergence of the feed pipe of 34½ in. of slurry used with the open-top cyclone is reasonably low. Furthermore, as no moving parts are involved, no power is required for rakes or motor-operated valves.

Summary

A different approach to gravitational classification equipment has been proposed to afford more *self-regulation* to the operation, enlarge capacity per square foot, and possibly increase sharpness of classification at no increase in power requirements. Experimental evidence indicates that this can be achieved by creating a free-vortex through the use of a tangential feed entry to a vessel somewhat similar to a conventional cyclone. However, no cover is used and vessel diameter will greatly exceed that of conventional cyclones. Results indicate also that the theory of this open-top cyclone will differ in many respects from that of the smaller pressurized cyclones. Some of this theory has been expressed from the experimental results obtained to date, but further work is necessary to a more complete fundamental understanding. The following conclusions are indicated at this time.

1—The open-top cyclone exhibits a mesh of separation and an equilibrium particle size well below the expected value for conventional gravitational classifiers on the basis of overflow capacity per square foot of pool area. Mesh of separation on 2.83 sp gr solids in these tests ranged from 100 to 140.

2—Mesh of separation and equilibrium particle size are considerably more immune to feed rate fluctuations than gravitational classifiers.

3—A peripheral weir discharge should not be utilized with an open-top cyclone if a minimum classification point is desired. Such a discharge arises from the lowest centrifugal force radius of the free vortex and simultaneously robs fluid energy from the vortex pattern.

4—A single, centrally originating overflow is desirable with a streamlined horizontal section to eliminate as much turbulence as possible. Furthermore, the open-top cyclone design should minimize all wall friction by maintaining smooth constant tapers in the conical section.

5—Slurry level above the feed pipe can be held to less than 3 ft, making energy requirements relatively unimportant. Possible energy equations have been proposed but remain to be obtained by further experimental data on different size units.

6—Overflow rates expressed as gallons per minute per square foot of settling pool as high as 37.3 have been obtained. This is 789 pct above the highest average of gravitational classifiers operating under similar conditions. It is believed that rates of 84 gpm per sq ft are possible by covering a larger percentage of cyclone diameter with inlet and overflow nozzles than were employed in the experimental unit.

7—Solid feed rates averaged 2.11 tons per hr per sq ft of settling area with a high of 3.36. This average is 349 pct above highest average for gravitational classifiers. These also could be increased 2.25 times, for the above reason.

8—Evidence points to higher classification efficiency with the open-top cyclone.

9—No plugging of the underflow spigot occurred and the open-top cyclone was operated with an overloaded discharge without injuring the mesh of separation.

10—Abrasion has been a difficult problem with conventional cyclones in certain installations. However, the open-top cyclone operates with much lower solid velocity at the underflow, where most of the conventional cyclone wear has occurred, owing to the low fluid forces involved. It is expected that abrasion problems will accordingly be decreased.

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Crushed Limestone Aggregates for Concrete

by Katharine Mather

This paper is an attempt to put together petrographic, physical, and chemical data about the large and varied group of rocks generally called limestones. Results of the properties of these rocks on their performance as concrete aggregates are discussed.

CRUSHED stone classified as limestone comprises about 70 pct of the crushed concrete aggregate and roadstone produced in this country, and the product has amounted in the recent past to about 97 million tons per year, valued at more than $123,000,000.[1] Therefore an evaluation of the properties of rocks classified in these statistics as limestones and an ability to predict their behavior as aggregate in portland cement concrete are matters of considerable economic consequence to producers and consumers of aggregate. This paper is an attempt to put together petrographic, physical, and chemical data about the large and varied group of rocks generally called limestones. It also points out certain results of the properties of these rocks on their performance as concrete aggregates.

The problems encountered in efforts to explain the behavior of limestones, dolomites, and related rocks as concrete aggregate lie on the frontiers between the ignorance of the geologists and the ignorance of the engineers. The mineral composition and textures of the rocks have not been adequately explored and described, and the effects of mineral composition and texture on the performance of aggregates have been worked out in only a few cases. There are instances of unsatisfactory performance and some generalizations of those experiences. There are many more examples of satisfactory performance than of unsatisfactory, but there is practically no published information as to why a particular aggregate was satisfactory in a specific use. There are all too few investigations of undeteriorated concrete; more is known about the pathology of concrete than about its normal structure and aspect.

As part of the investigation of aggregates, the classification of rocks is not an end in itself, but is a step toward recognition of properties expected to influence the behavior of aggregates in their intended use. Rock of any type may perform well or poorly as concrete aggregate, depending on the physical condition of the rock, its physical and chemical properties as compared to those of the matrix in which it is placed, and the circumstances under which the concrete is exposed. Some classification and description of the limestones is necessary to distinguish subgroups whose members will probably behave in similar ways and to evaluate the group as a whole.

The group name, limestone, includes and tends to conceal the identity of rocks that have a very great range in chemical and mineralogical composition, structure and texture, conditions of deposition, subsequent history, and suitability for use as concrete aggregate. All the rocks classified as limestones include one or both of the most abundant rock-forming carbonate minerals, calcite and dolomite, as major constituents. *Carbonate rocks* is a more descriptive and more accurate group name than limestone, which has generally been used to include dolomitic or magnesian limestones and dolomites as well as calcitic limestones.

Classification and Description of the Carbonate Rocks

The question of names to be applied to the carbonate rocks may appear academic, but it is of practical importance as long as there are specifications for aggregate that result in acceptance or rejection of rocks by name and as long as failure to make distinctions among the carbonate rocks leads to misunderstandings of their properties and of what may be expected of them in performance. Materials included in production statistics as limestone and tested for concrete aggregate as limestone range in fact from calcareous shale and calcareous or dolomitic sandstone and argillaceous limestone and dolomite to calcitic limestone and pure dolomite and to rocks close to calcitic and dolomitic marble. To take a simple example: consider two manufactured coarse aggregates, both dense and unweathered, one composed principally of crystalline calcite and the other of crystalline dolomite, but both known as limestone. If each is used as coarse aggregate in a standardized concrete mixture with the same fine aggregate used in both cases, made into concrete specimens, and tested in accelerated freezing-and-thawing, the test results will probably be quite different, because of the differences in internal bond and thermal properties between the two aggregates. As long as both aggregates are called limestone, the test results cannot be explained, or used except in a very limited way; if the aggregates are distinguished as limestone and dolomite and the consequent differences in texture and thermal properties are recognized, the results are reasonable.

Classification of the carbonate rocks used as aggregate should be based on structure and on composi-

K. MATHER, Member AIME, is Chief, Petrography Section, Concrete Research Division, Waterways Experiment Station, Corps of Engineers, U.S. Army, Jackson, Miss.

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tion and texture. These last two are fundamental properties[2] which can be related to aggregate performance. The relation of mineral composition to performance is quite often clear and easily recognizable. Less attention has been paid to texture, probably because it is hard to describe, almost infinitely varied in the carbonate rocks, and often difficult to explain. Structure, particularly bedding, cleavage, and jointing should be considered as far as the samples or exposures permit because of its influence on particle shape and sometimes on the maximum size that can be produced.

The important groups of constituents to be taken into account are the carbonate minerals, the detrital constituents other than clay, the clay minerals, and the remaining, largely secondary constituents. Ordinarily either calcite or dolomite or both are the carbonates present, but others, for example siderite, are possible. It may be important to know the identity of the carbonate or to recognize the presence of both calcite and dolomite and the way they are distributed, since there are differences in thermal properties between them and differences in texture and interlock between calcitic limestones, dolomitic limestones, and dolomite. These in turn cause differences in pore structure and internal bond. The most probable detrital constituents are quartz, chert, and feldspar; various ferromagnesian minerals and opal have been found. Some of the clay minerals can have very marked effects on aggregate performance even when they are present in very small amounts, depending on their type, distribution, and accessibility to water. The remaining constituents ordinarily found are secondary chert or quartz; sulphides (pyrite and marcasite), sulphates (anhydrite, gypsum), phosphates (carbonate apatites), and hydrocarbons are among the possibilities that have been encountered.

In practice these four generalized groups may often be simplified; many carbonate aggregates can be described in terms of variations among four end members, calcite, dolomite, sand or silt, and clay, or among three of those four. Fig. 1 shows a tetrahedral diagram relating the four components, as well as subdivisions of the four faces and rock names regarded as appropriate. Compositions may be read along the edges of each triangle. The diagrams are based primarily on discussions by Pettijohn.[3]

Description of texture should include measures of grain-size if the rock is even-grained or information about the size-ranges of the groups of constituents (the larger clastic grains, the matrix, and the cement), and about the arrangement of each group with respect to the others. Most of the textural terms current about sedimentary rocks are inadequately defined and unfamiliar to most geologists and most engineers. Their use is likely to cause confusion and annoyance. Krynine[7] has provided a very useful discussion on how to describe texture. Sander's examples[4] are useful and interesting but written in highly specialized language.

No new names or new classification for carbonate rocks are proposed here. The writer's experiences in trying to fit aggregate samples into classifications have led her to believe that none of the detailed classifications with which she is familiar are satisfactory for this use. The basic standardized descriptive scheme used for igneous rocks can be applied to the limestones and dolomites, and the names constructed will consist of the following parts: color, textural modifier, varietal minerals and cement, and principal name.[2] All four of the generalized groups of constituents, the carbonates, detrital constituents other than clay, the clay minerals, and the secondary products, are important in assessing the potentialities of a carbonate aggregate. The texture is the second important consideration, including the ranges in grain size or crystal size of each group of constituents and their arrangement with respect to each other. The third is structure, particularly bedding, cleavage, jointing, and alternations of different lithological types. Attention to these three considerations can result in rock names that are clear, adequately defined, and objectively based. These descriptions will contain information permitting as good predictions about performance as can now be made.

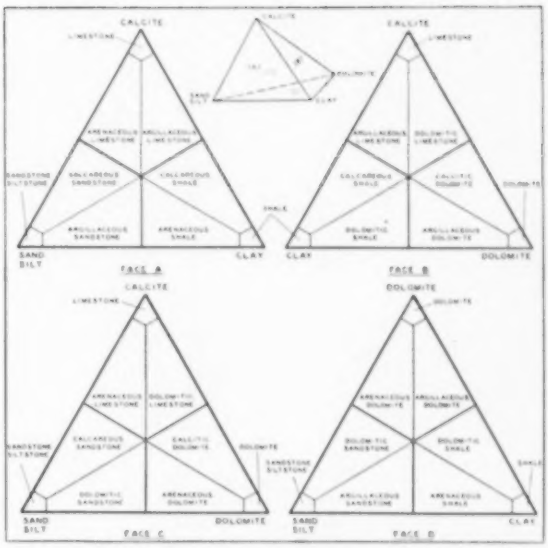

Fig. 1—Tetrahedral diagram showing variations in composition among rocks composed of calcite, dolomite, clay, and sand-silt.

Properties of Carbonate Rocks as Concrete Aggregate

It is convenient to discuss the effects of rock properties under two headings: those that influence production and those that influence use in concrete. The same properties are involved, but some of those influencing production act at that stage to eliminate potential sources, and some that make minor or negligible differences in production have marked effects in concrete.

Properties Influencing Production: To be produced as concrete aggregate, a carbonate rock must be available in adequate quantity for the intended use, must be present in beds thick enough to permit aggregate of the desired maximum size to be made, and must be strong and cohesive enough to be processed and emerge as a graded product in the size-ranges desired. The second requirement eliminates some thin-bedded deposits as sources of aggregate for mass concrete, although they are suitable for pavements or buildings. The third eliminates many of the consolidated but soft Cretaceous and Tertiary limestones of the Gulf Coast that crumble when they are crushed.

Interbedded shales in carbonate rocks give rise to problems of several kinds in both production and use. One such problem was solved by the information obtained from petrographic investigations

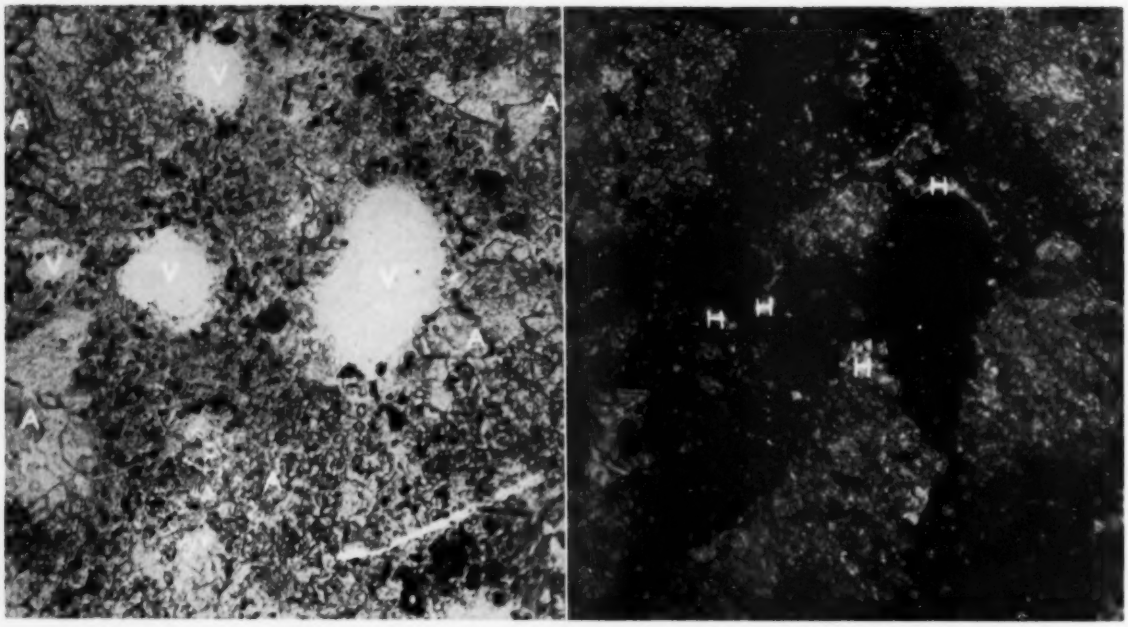

Fig. 2—Dam B, interior mass concrete containing limestone aggregate, X165, plane light (left), and crossed nicols (right). At left are shown voids (V), aggregate (A), and scattered dark residual cement grains. At right, calcium hydroxide crystals (H) form fairly small areas. The multitude of minute light dots in the groundmass are limestone dust smaller than 15 microns.

and laboratory studies of processing of aggregate for a dam in southern Kansas located about 40 miles from the nearest commercial quarry. A 12-ft section of limestone was exposed near the dam site. Blocks of ledge rock from each bed were shipped to the Waterways Experiment Station, with enough information to permit complete reconstruction of the section. The limestone was medium-grained and somewhat clayey, with open bedding planes at 4-in. to 4-ft intervals, and the rock above and below each open plane was porous and stained for thicknesses of ¼ to 1 in. Stylolites were spaced at 2-in. to 1-ft intervals; the rock broke easily on some of the stylolites but on others the bond was strong. Some of the stylolites contained soft fragile black shale which broke away easily from the limestone and disintegrated in water. Shale and porous weathered rock amounted to 10 pct of the total thickness. It appeared that much of the shale and much of the weathered porous limestone could be eliminated by crushing and washing, leaving uncoated limestone of satisfactory quality. Aggregate processed in the laboratory, made into concrete, and tested in accelerated freezing and thawing confirmed these indications. The aggregate plant installed at the site included scrubbers, and satisfactory aggregate was prepared from an apparently unsatisfactory source.

Properties Influencing Use: Performance of any concrete aggregate depends on its physical condition, its physical and chemical properties compared to those of the matrix in which it is placed, and the circumstances under which the concrete is exposed. Great variations may therefore be expected in performance of any one coarse aggregate if it is used in combination with several fine aggregates, in different structures, different concrete mixtures, and different exposures. These variations in performance are possible even if the aggregate in question is as uniform as a quarried stone can be; if the aggregate varies considerably in mineral composition and texture, as is often the case, the situation is more complicated. Considerations such as these render meaningless the attempts to classify aggregates as *good* and *bad* or *satisfactory* and *unsatisfactory* except with regard to a specific use or specific set of requirements. The same properties that make an aggregate highly serviceable in building construction or pavement may make it less desirable in mass concrete, and vice versa.

One of the properties of carbonate aggregates that varies with variation in mineral composition is linear coefficient of thermal expansion. The average linear coefficient of thermal expansion of the mineral, calcite, is 2.9×10^{-6} in. per in. per degree F. The average for dolomite is 4.2×10^{-6}. Relatively pure calcitic limestones usually have thermal coefficients in the range of 2 to 3×10^{-6}; relatively pure dolomites have coefficients between 4 and 5×10^{-6}.[5-7] Dolomitic limestones have coefficients of intermediate value. The average value for quartz is about 6×10^{-6} and that for chert a little lower, so that the presence of either one in appreciable quantities will raise the average thermal coefficient of either a limestone or a dolomite. A few failures of concrete ascribed to coarse aggregate of very low thermal coefficient or to anisotropic thermal expansion of coarse aggregate have been reported.[8, 9]

Callan[6] has reported on the relations found between durability of concretes as measured by accelerated freezing and thawing of concrete beams, difference in average thermal coefficient of coarse aggregate and mortar (Δc), and absorption of coarse aggregate. Seventy-eight aggregate combinations were studied. There were 22 coarse aggregates of which 19 were calcitic limestones, and 29 fine aggregates, 18 of them manufactured limestone sands. Most of the rest were natural sands high in quartz. Each coarse aggregate had been tested in concrete in combination with several different fine aggregates. The concrete combinations made from selected

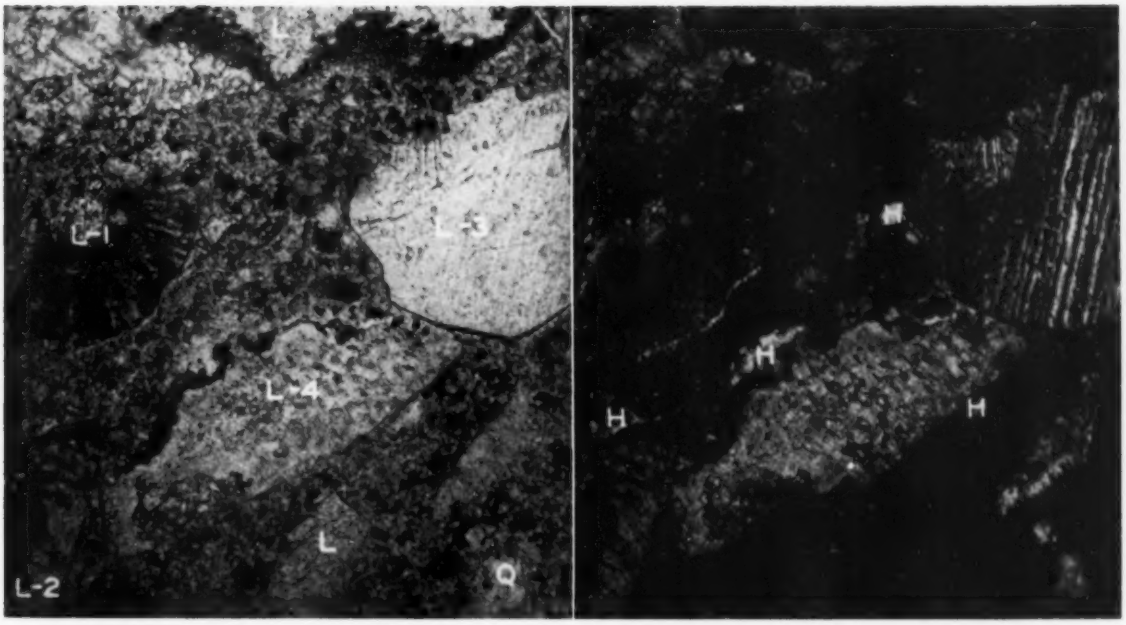

Fig. 3—Dam E, interior mass concrete, X165, plane light (left) and crossed nicols (right). At left, lower limestone fine aggregate (L) shows rhombohedral cleavage; L-3 shows twinning in plane light and with crossed nicols; quartz (Q). At right, calcium hydroxide is marked H; it also forms narrow pale gray borders along the lower edges of grains L-1 and L-3. The scattered limestone dust is smaller than in Fig. 2.

aggregates were highly standardized. All were made from Type 2 moderate-heat cement. All had a constant water-cement ratio of 5.5 gal per bag of cement, an air content of 4.5 ± 0.5 pct, and both coarse and fine aggregate of specified grading, with 1-in. maximum size stone and a ratio of 36 to 42 pct sand to total aggregate by volume. The cement content was adjusted to give the desired consistency but normally fell between 5 and 6 bags per cu yd. From each concrete mixture, three 3½x4½x16-in. beams were molded, cured for 9 days at 70°F and 100 pct relative humidity, and tested in an automatic freezing and thawing apparatus[10] for 300 cycles, or until the average relative dynamic modulus reached 50 pct. The average thermal coefficient of each coarse aggregate and of each mortar was determined; standard physical tests were made on all the coarse and fine aggregates. Statistical analyses of the test results indicated that in this series of highly controlled concretes tested in standardized conditions, there was a multiple correlation between durability (as measured by the freezing and thawing test), difference in thermal coefficient between coarse aggregate and mortar, and absorption of the coarse aggregate. The coefficient of multiple correlation for this relation was 0.72; the coefficient of partial correlation between durability factor and difference in thermal coefficient was 0.60, and that between durability factor and absorption of coarse aggregate was 0.47. Differences in thermal coefficients between coarse aggregate and mortar were thus more important in these tests than differences in absorption.

Another investigation made at the Waterways Experiment Station provides some information on the relation of type of clay in limestone to the durability of concrete as measured by accelerated freezing and thawing tests.[11] This series of tests included nine carbonate coarse aggregates, seven of them calcitic limestones, one dolomite, and one dolomitic limestone that contained considerable sandy limestone and calcareous sandstone. Standard physical tests and thorough petrographic examinations were made of these aggregates. Each was combined with manufactured sand from the same source, made into concrete, and tested in accelerated freezing and thawing. The coefficients of thermal expansion of each coarse aggregate and each mortar were determined. The petrographic examinations included determinations of the percentage insoluble in hydrochloric acid in each lithologic type in each sample and of the mineral composition of the insoluble residues, and micrometric determinations of the percentage of cracks and voids in each major lithologic type in each sample.

The relation between durability factor and difference in thermal coefficient between coarse aggregate and mortar that had been found in the investigation reported by Callan was also apparent here, but other relations became clear when freezing and thawing results, thermal coefficients, petrographic data, and absorption were all considered. The clay minerals found in five of the nine aggregates belonged to the kaolin group and amounted to 1 to 5 pct of the total. The other four contained montmorillonoid clays, capable of swelling; in one, the percentage was negligible; in two, there was 4 pct each, and in the last, there was 8 pct. Only one of the aggregate combinations in which the clay was kaolin had a durability factor below 80, and that aggregate had fairly high absorption and contained 15 pct cracks and voids. The aggregate that contained a trace of montmorillonoid clay had low absorption and gave a high durability factor. The 4 pct of swelling clay in each of two samples was concentrated in one porous rock type in each sample. Both gave durability factors of 75 or less. The fourth was a dolomite with 8 pct of swelling clay disseminated

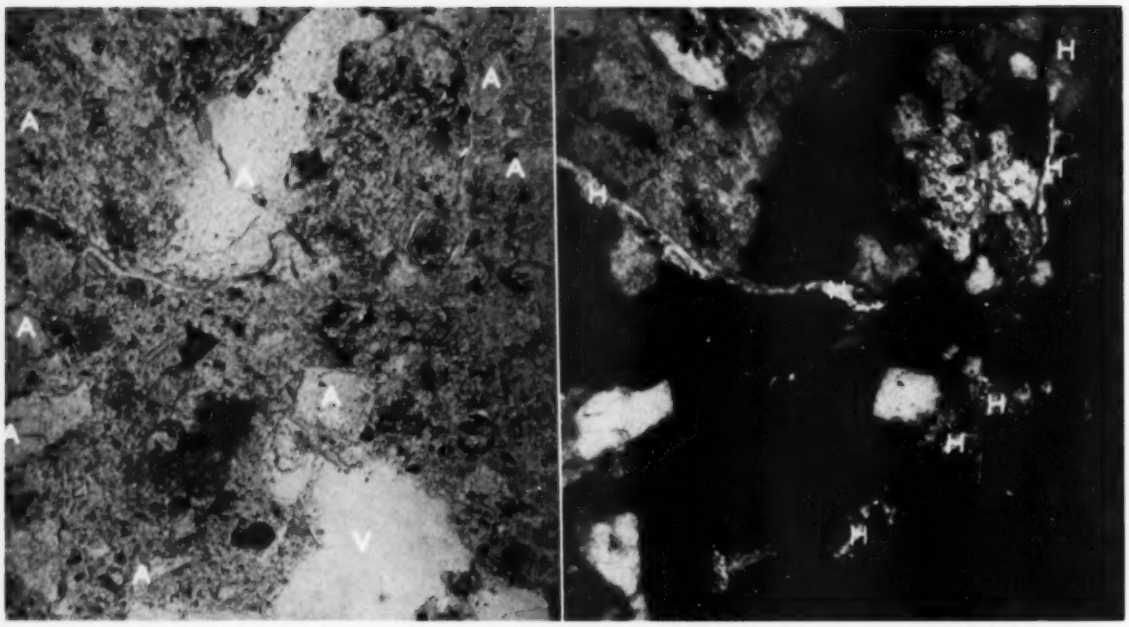

Fig. 4—Dam D, interior mass concrete containing diorite aggregate, X165, plane light (left), and crossed nicols (right). At left are shown voids (V), aggregate (A), and dark residual cement grains. In the right photomicrograph, the calcium hydroxide crystals are large, there is very little aggregate smaller than 25 microns, and there are larger areas of uninterrupted gel. Compare with Figs. 2 and 3 for differences with limestone aggregate concrete.

in dense rock; it gave a durability factor between 75 and 80.

While there were not enough aggregate combinations tested to make correlation analysis useful, the degree of association between type of clay mineral and durability factors above and below 80 could be computed by use of the chi-square test.[17] This calculation indicated that the probability of chance occurrence of the association between swelling clay and durability factor below 80 was less than 2 in 100.

Several examples of failure or distress in service of argillaceous limestone aggregates in which the major factors appear to have been the presence of swelling clay in limestones of high absorption were described by G. F. Loughlin.[18] He investigated a number of limestones from Pennsylvania, Illinois, Iowa, and Kansas which had either poor service records or peculiar test results. All these rocks were argillaceous limestones, and all were shown to contain more than 5 pct of montmorillonoid clays. Variations in amount of clay, size and distribution of clay pellets, absorption, and distribution of permeable bedding planes and fractures appeared to explain the differences in performance between rock from different beds in the same quarry and from quarry to quarry. Small differences in amount and arrangement of clay accounted for differences in test results and in service.

The data so far available indicate that 5 pct or less of kaolin clay in dense limestone may not injure the performance of the limestone as aggregate, but that similar percentages of montmorillonoid clays in limestone with 1.5 to 2 pct absorption may produce a noticeable effect on durability as determined by accelerated freezing-and-thawing of concrete, and 10 to 20 pct of montmorillonoid clay in limestones which have 2 pct or more absorption, used in pavement laid on poorly drained foundations, may cause the pavement to fail. No information has been located about the effects of some of the other clay minerals (illites, chlorites, attapulgite) that are recorded or possible constituents of carbonate aggregates. Information is also lacking on the mineral composition of shales interbedded with limestones, as that composition affects the performance of the aggregate. Not all these shales are eliminated in aggregate processing but persist especially in the smaller sizes of the finished coarse aggregate. Some cause pop-outs in freezing and thawing test beams and in concrete structures, but some do not.

Dolomitic limestones and dolomites have not been as well represented as limestones among the carbonate aggregates tested at the Waterways Experiment Station. One group of six dense calcitic dolomites was examined and tested during an aggregate survey made for a dam in Virginia. Four of the six were quite pure calcitic dolomites. One contained some cherty and some sandy dolomite as well. One was argillaceous dolomite, but the clay was not identified. All had less than 1 pct absorption. When each was tested in freezing-and-thawing of concrete in a mixture containing manufactured sand from the same source, all had durability factors of 85 or more, except one pure sugary-textured highly fractured dolomite.

Secondary silica in the form of chert or quartz or opal may affect both production and use of a carbonate aggregate. Vein quartz, like detrital quartz, increases the resistance to crushing of a carbonate rock and raises its thermal coefficient. Chert has a similar influence on crushing characteristics, particle shape, and thermal properties, and in addition is sometimes potentially reactive with the minor alkalies in portland cement. Opal is apparently much less common than chert, but opal is the major constituent of the Californian rock usually called *siliceous magnesian limestone* in the literaure of alkali

reaction,[14] a dolomitic opaline shale like that forming large concretions in the Monterey formation.[15] This peculiar rock is unique. It is distributed in a restricted area and is not what is ordinarily called magnesian or dolomitic limestone. Three railroad structures in southern Ohio and Indiana containing limestone and dolomitic limestone coarse aggregate and natural fine aggregate are believed to have deteriorated because of reaction between chalcedonic chert and opal, as constituents of the fine aggregates, and the minor alkalies in the cement.[16] The limestone coarse aggregates are not involved. Two of the structures were built in 1901. The age of the third was not given, but the youngest structures examined in this group of 13 were built in 1927. Each had some features suggesting poor construction practice, poor design, or poor drainage.

Four additional examples of alkali-aggregate reaction in field concrete containing limestone or dolomite as coarse aggregate have been discovered or confirmed as the result of the examination of samples by the writer. They have common factors suggesting that the alkali-aggregate reaction in these cases may be a secondary consequence of other deterioration in the concrete, rather than a major or a primary cause of deterioration. In three cases chalcedonic chert and glassy volcanic rock fragments, constituents of the fine aggregates, were the reactive ingredients, and there were no signs of reaction by the coarse aggregate. In the fourth, chalcedonic chert nodules in the limestone coarse aggregate had reacted, but this was the most porous and permeable concrete ever examined at the Waterways Experiment Station, and it was exposed to severe freezing-and-thawing and alternate wetting and drying. All the structures were in regions with fairly severe winters, and all were non-air-entrained concrete. No distress was recognized until the structures were more than 15 years old, and there was evidence of poor construction practice or high water-cement ratio or both in three. All contained smooth-surfaced rounded natural sands high in quartz.

The examples so far discussed to illustrate effects of variations in properties of carbonate aggregates with changes in composition have been concerned with variations among several groups of carbonate aggregates: limestones and dolomites, argillaceous limestones and dolomites, and cherty limestones. The last group of examples will illustrate variations in the properties of concrete resulting from differences between some carbonate aggregates and some non-carbonate aggregates.

In 1952 the writer had the opportunity of examining 10-in. diam diamond drilled cores of interior and exterior mass concrete from five Corps of Engineers dams, as part of a study of causes and methods of controlling cracking in mass concrete structures.

The fine and coarse aggregate used in the concrete in three of the dams was manufactured from limestone. In one it was manufactured from granodiorite, and in the fifth it was manufactured from quartz diorite. Each dam was represented by exterior and interior cores, and information was provided on the location in the structure from which the cores were taken, the age of the concrete when the cores were drilled, and the mixture proportions and aggregate gradations. Four of the aggregates had previously been tested at the Waterways Experiment Station, so that results of the standard physical tests, of freezing-and-thawing tests of concrete specimens, and of petrographic examinations

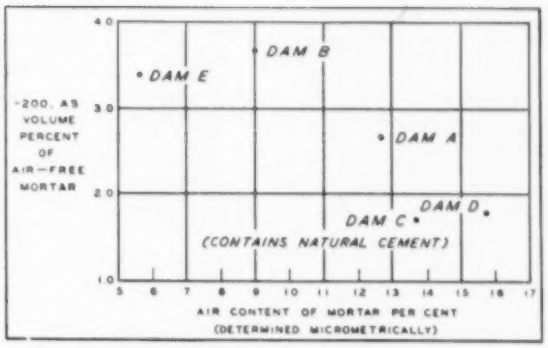

Fig. 5—Relation of average air content of mortar and amount of material finer than No. 200 sieve in aggregate.

were available. All the cores were examined macroscopically, the coarse aggregate content and air content of the hardened concretes were determined, sawed and broken surfaces were examined at low magnifications, and many thin sections of the concrete were prepared and examined.

The three limestone aggregates were not extreme variants in the limestone group, but they did differ considerably in structure, texture, and composition. One was relatively pure calcitic limestone, on the average finer-grained and more even-grained than the other two. The second was oolitic and fossiliferous limestone grading into sandy and shaly limestone. It ranged in grain size from very fine-grained to coarsely crystalline. The third contained about 25 pct chert, was the most cohesive of the three, and was moderately fine-grained. The two igneous aggregates were medium-grained slightly foliated granodiorite and moderately coarse-grained quartz diorite. The granodiorite was not altered but had some internal microfractures. The diorite was altered but was thoroughly interlocked by intergrowths of feldspar and quartz in the groundmass.

Comparisons of thin sections of the concretes made with limestone aggregates and those made with siliceous aggregates revealed several conspicuous differences between them. One was the difference in crystal size of the calcium hydroxide, which is one of the products of cement hydration, between the concretes with limestone aggregate, Figs. 2 and 3, and the concretes with siliceous aggregate, Fig. 4. Those with limestone aggregate had relatively small calcium hydroxide crystals, and those with siliceous aggregates, relatively large ones. This general relation, observed in all these cores, has also characterized all the comparisons made so far between undeteriorated concretes with limestone aggregates and comparable concretes with siliceous aggregates. It was also found between concretes of the same age made with the same cement and what was intended to be the same gradation of fine aggregate. The second consistent difference between the concretes with limestone aggregate and the concretes with siliceous aggregates was the difference in amount and particle size distribution of material finer than the No. 200 sieve in the aggregates. (Compare Figs. 2 and 3 with Fig. 4.) The difference in crystal size of the calcium hydroxide and the difference in abundance and size of the aggregate passing No. 200 implies another point illustrated by the photomicrographs: there are larger areas of apparently amorphous hydrated cement gel uninterrupted by crystalline particles in the concrete with siliceous aggregates.

This observation correlated with the tendency to crack in the cores from which the sections were made. When sawed surfaces of the concretes were examined at low maghification after several months exposure to air, all the limestone-aggregate concretes had more cracks per 100 sq in. than either of the siliceous-aggregate concretes.

The consistent difference in amount and particle-size distribution of the aggregate finer than the No. 200 sieve depends on differences in physical properties between the mineral constituents in the two groups of aggregates. The percentages passing No. 200 in the fine aggregate gradations reported showed an average of 3.5 pct in the two containing siliceous aggregates, and 4.8 to 8.1 pct in the three containing limestone. The actual quantity of −200 in the concretes included the reported amount, as well as an additional amount consisting both of that adhering to coarse aggregate and not allowed for and that produced by wear between the point where samples were taken for sieve analysis and the final location of the concrete.

In the crushing of brittle crystalline solids, the amount of new surface formed per unit of energy input increases as the hardness on Moh's scale decreases.[17] Quartz has a hardness of 7; the feldspars range from 6 to 6.5; calcite has a hardness of 3. The same expenditure of energy therefore yields about 1.8 times as much new surface on calcite as on quartz and feldspar.[17] If all the aggregates were similar in original grain size and interlock and were subjected to the same amount of abrasion during batching and mixing, the amount of new surface produced on the limestones should be about 1.8 times that produced on the granodiorite and quartz diorite, and the amounts of −200 mesh should be proportionately different. As a matter of fact, the limestones were of finer grain; it was difficult to compare the degree of interlock between the limestones and the diorite and granodiorite.

There were two other features of these concretes related to the amount of −200 in the concretes. Like all portland cement concrete placed by conventional methods, all these cores showed some evidence that the mortar had bled water and subsided under the larger coarse aggregate before set. However, the limestone-aggregate concrete bled less and subsided less under the aggregate than the siliceous-aggregate concretes. These observations may be explained as due to the effect of additions of material of high specific surface in reducing bleeding, and the greater effectiveness in this respect of pulverized limestone as compared to pulverized quartz of similar specific surface.[18]

The additional effect of the larger amount and smaller average particle size of the −200 limestone was a reduction in amount of entrained air in mortar, Fig. 5, and a reduction in maximum and median size of the individual air voids. A given air content in voids of small size should give more frost resistance than the same air content in voids of larger size.[19]

Summary

It is believed that the examples that have been presented illustrate the following points:

1—The carbonate aggregates have a wide range in composition, from calcitic limestones and pure dolomites to argillaceous and arenaceous limestones and dolomites. Each of these types may also vary widely in texture and structure.

2—The variations in composition, texture, and structure result in wide variations in physical and chemical properties, and potential or actual suitability for use as concrete aggregate.

3—Carbonate aggregates that contain even small amounts of swelling clay and that also have fairly high absorption may give poor service, particularly in concrete exposed to severe weathering or frequent wetting and drying. However, dense argillaceous limestones in which the clay is not swelling clay may give good service. Determination of the clay mineral or minerals and the amounts in which they are present offers the most satisfactory basis for a decision.

4—Calcite and dolomite, the major carbonate minerals, are considerably softer than the feldspars, quartz, and the ferromagnesian minerals that are the principal constituents of most of the other sorts of rocks used to produce crushed aggregates for concrete. As a consequence, more material passing the No. 200 sieve is likely to be produced in processing and handling carbonate rocks than would be produced from rocks of other types.

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Recharging Ground Water Reservoirs With Wells and Basins

by M. L. Brashears

IN the last 15 years industrial use of ground water has more than doubled, and in 1951 amounted to 5 billion gallons per day. A similar sharp increase in the utilization of ground water for irrigation and public-water supply occurred in the same period. In many areas rapid increase in withdrawal from wells has taken place almost entirely unhampered by regulatory control and with little or no integration of effort. As might be expected, the chief interest in many regions has been maximum production rather than sustained perennial yield. As a result, widespread depletion of underground reservoirs and deterioration of the quality of the water stored in them has taken place in many areas, even though total pumpage in the United States is far below ultimate potential. Of even more concern is the fact that excessive withdrawal has drawn salt water into the reservoirs beneath many heavily populated centers along the Atlantic, Gulf, and Pacific coasts, causing costly abandonment of pumping plants.[1]

Many hydrologists expect that consumption of water will rise rapidly in the near future, and some predict that industrial requirements will more than double in the next decade.[2, 3] Thus it appears likely that the draft on many already heavily pumped underground reservoirs will be greatly increased and the search for additional sources of usable ground water intensified in years to come. In view of this, industry as a whole will be forced more and more to recognize the potentialities and limitations of ground-water reservoirs and to utilize them more effectively to prevent costly water shortages and disruption of production. Through painful experience, some industries are already well aware of the need for effective water utilization, and have managed individually or through joint effort to check trends threatening to deplete underground reservoirs completely or to impair the quality of the water. Various remedial measures have been used to bring about successful management of local or regional ground-water resources. Of these, replenishment of aquifers by recharge wells or basins has played an important role in overcoming some ground water problems.

Artificial recharge of underground reservoirs by water spreading has been practiced successfully in the United States for many years. In the West it has become an important method of salvaging flood run-off for irrigation of crops and maintenance of public water-supply reserves, and it is used to some extent in parts of the East. Artificial recharge by means of wells, on the other hand, is a relatively new development. Until recently it was employed in only a few areas, principally along the East coast. For the last few years, in the ever increasing search for additional water supplies, industry has had greater recourse to this method. Utilization of recharge wells to control the temperature and quality of underground water supplies is also being considered seriously.

Operation of recharge wells, like water spreading, is governed largely by local conditions. It requires water relatively low in turbidity, whereas in some areas water spreading has been used successfully with water of high turbidity and silt content. However, water spreading must be employed in large areas and can be carried on effectively only where aquifers crop out at the surface. Recharge wells can be used in limited space.

Recharge wells are similar to production wells except that the water flows in the opposite direction. Thus any water-bearing bed that will yield water to wells may be recharged by wells. Often, however, the water available for recharge is of a different character and temperature from that existing in the ground-water reservoir and if transmitted directly underground from a recharge well to a production well might require expensive or difficult treatment before it could be used.

Fortunately the physical characteristics of reservoir beds, which control the movement and behavior of ground water, are generally not homogeneous. Moreover, the movement of ground water is very slow because of the frictional resistance of the reservoir beds. By taking full advantage of hydrologic and geologic conditions, it is therefore possible in many instances to bring about favorable changes of temperature and dilution as the water moves from the recharge wells underground to the production wells. Furthermore, if the natural quality or temperature of ground water is unfavorable for industrial purposes, recharge wells may be used to introduce water of more favorable quality or temperature into the ground-water reservoir.

When water is discharged into a recharge well, the head in the well is increased. Because of this, a cone of elevation is produced on the water table or the artesian pressure surface in the area surrounding the well. The cone of elevation is similar to the cone of depression produced around a pumping well except that the apex of the cone is above the water table or artesian pressure surface. Thus if a recharge well and a production well tapping the same water-bearing bed are close together, as would be the case at many industrial plants, some of the water discharged from the recharge well would be drawn into the production well within a short time. Under such conditions it is apparent that water of unfavorable temperature and chemical characteristics should not be used for recharging.

The more important ground-water reservoirs in the United States often consist of alternating layers of impermeable beds and porous material that will yield water readily to wells. Physical characteristics of individual beds in a ground-water reservoir may not persist over great distances, the impermeable layers grading into beds that will yield large quantities of water. Thus the water-yielding material in underground reservoirs, whether large or small,

M. L. BRASHEARS is associated with Leggette and Brashears, New York City.

Discussion on this paper, TP 3609H, may be sent (2 copies) to AIME before Dec. 31, 1953. Manuscript, January 6, 1953. Chicago Meeting, October 1952.

often consists of a maze of interfingering and overlapping channels of permeable material of varying thickness through which ground water moves on its journey from intake area to discharge area, or from recharge well to production well.

Because of this irregularity in most ground water reservoirs it is often possible to pump water from one of several water-bearing beds and to recharge another bed separated from it hydraulically, so that movement of water between the two wells is prevented. Movement of ground water is influenced by every minute irregularity of the reservoir beds. Even thin impervious beds or variations in permeability within an individual water-bearing bed may greatly influence the direction of movement of ground water. Deposits of sand and gravel are more or less horizontally stratified. This often results in very thin layers of fine sand, silt, or clay, the presence of which would not be recognized during the drilling of a well. The vertical permeability of deposits of this kind is much less than the horizontal permeability, the thin layers of relatively impervious material greatly retarding the vertical movement of water. With such deposits there often is a great advantage in having the recharge well of different depth from that of the production well.

All water-bearing beds serve not only as reservoirs but also as conduits to carry water to wells from the areas where it enters the beds. In this respect ground-water reservoirs differ from lakes and surface reservoirs in that they offer great resistance to movement of water through them. Serious water shortages may develop in ground-water reservoirs not only because of insufficient storage capacity or natural replenishment but also because the slow movement of ground water through the reservoir beds, even under high hydraulic gradients, cannot meet the demand of pumping wells. This important characteristic of water-bearing beds is important in all problems of artificial recharge.

No two ground-water problems are exactly alike, and each may result from a complex set of natural and man-made conditions. In some instances it may be possible to remedy the situation by means of artificial recharge, but in others artificial recharge may cause additional problems. Careful evaluation of conditions causing the problem should be made if waste of time and money is to be avoided. Spacing of wells is of major importance. Where wells are closely spaced, as in highly developed areas, artificial recharge practiced at only one industrial plant may produce only minor beneficial results. Joint effort, however, may bring about a satisfactory remedy for all. On the other hand, where wells are spaced great distances apart the entire benefits of artificial recharge may be realized by the operator.

Broadly speaking, industry may obtain water for artificial recharge from the ground-water reservoir being recharged or from some outside source. The latter may be derived from a municipal supply system, nearby lakes and streams, or even from wells tapping another ground-water reservoir. It has been demonstrated on Long Island and elsewhere that if properly treated, industrial waste water, storm water, and sanitary sewage may be used for recharging. Selection of source would be governed by many factors, chief of which would be temperature, turbidity, and chemical character of the water and the distance to the source.

There are numerous situations where artificial recharge wells could be profitably used by industry for cooling large quantities of water during the warmer months or for controlling temperature and chemical character of water in the underground reservoir. Under some conditions, recharge wells may be used profitably for disposal of waste water and the draining of water-logged land.

Usually the amount of water stored in an underground reservoir is equal to annual replenishment during several years or more. Because of this, water may often be pumped for short periods of time at rates considerably greater than the rate of perennial replenishment without creating serious problems. However, each period of heavy withdrawal must be followed by a period of little or no pumping, or the water must be replaced by artificial recharge. Such water-management procedure has been carried out successfully at several places in the United States. Perhaps one of the most striking examples of the use of artificial recharge to prevent exhaustion of available supply took place during World War II in Louisville, Ky, where an acute water shortage developed because of excessive withdrawal. Several industrial well fields were pumped at a rate of 60 million gallons daily, about twice the amount of natural replenishment. Late in 1943 the yield of some wells had dropped far below requirements and prompt action was needed to prevent further decrease in yields and to maintain an adequate supply of water with a summer temperature no higher than 60°F. Through joint effort, the industries arranged to use Louisville municipal water for several months during the winter and spring of 1944. During the same period the depleted underground reservoir was recharged by the pouring of cold water from the city supply into industrial wells at the rate of about 1000 gal per min. According to Guyton[4] this increased the potential yield of the area during the entire summer of 1944 by about 1000 gal per min and curtailment of production was prevented.

Similar beneficial results have been obtained at an industrial plant in the Binghamton, N. Y., area, where gradual increase in pumping from a small underground reservoir at the plant site caused a large decline of water levels. Additional wells were drilled several thousand feet from the plant near the Susquehanna River, the major source of the ground water. It was soon observed that although there was a lag of several months the temperature of the water pumped from the newer wells was governed by changes in temperature of the Susquehanna River.

During the late summer and early fall, temperature of the water from the newer wells rose above the operating maximum of 55°F. A schedule of recharging has remedied this situation. Cold water taken from the wells near the river during the off-peak season is pumped into the cone of depression around the plant, where it is held in storage until needed the following season. Under similar conditions elsewhere it is probable that cold water pumped from off-site wells or streams could be transported economically over long distances.

Canton, Ohio, affords another example of the successful operation of recharge wells in utilizing an underground reservoir for temporary storage of excess run-off. According to Kazmann[5] one of the Canton well fields lies over a buried valley containing two gravel aquifers separated by a layer of clay. The upper aquifer, although readily recharged by streams which cross it, is relatively thin and has

insufficient storage capacity to furnish the desired supply during periods of low stream flow. The lower aquifer, although comparatively thick, has no clearly determinable source of recharge and would be quickly exhausted if pumped heavily. Three horizontal collector wells were sunk along streams crossing the area. These collectors tap both aquifers and allow water to flow continuously from the upper to the lower, where it is held in storage until needed. The combined rate of recharge of the three collectors is 8 million gallons per day. One of the collectors is equipped with pumps and produces 10 million gallons per day from the lower aquifer.

As a result of control by the state government, more recharge wells have been drilled on Long Island, N. Y., than in any comparable area in the country. In 1933 it was discovered that excessive pumping in the previous decade or two had lowered the water table below sea level in an area of more than 40 square miles in Brooklyn and that sea water was moving into this area. Rough estimates indicate that approximately 100 billion gallons of ground water were mined during the period in which pumpage exceeded natural replenishment.

To protect public water supplies on Long Island from further over-development the state legislature passed a conservation law requiring the approval of the State Water Power and Control Commission before a well was drilled having a capacity greater than 100,000 gal per day. Since passage of the law the Commission has required that water pumped from new cooling and air conditioning wells be returned to the aquifer from which it it withdrawn. More than 300 recharge wells have been installed since 1933.[6] About 10 pct of these are industrial wells operated the year round. The remainder are operated only in the warmer months of the year.

The experience gained on Long Island during the last 20 years clearly shows that recharge wells can be operated successfully in unconsolidated deposits (the most productive aquifers in the heavily industrialized areas of the nation) under almost all geologic and hydrologic conditions. However, construction methods must be proper for the physical characteristics of the aquifers and the water used for recharging should be relatively clear. Moreover, hydrologic conditions at some locations interpose limitations. For instance, one limiting factor is depth of the water table below surface. If the water table is near surface the build-up of head in the recharge well must be relatively small. Under such conditions it is generally possible to create a drawdown in a pumping well greater than the maximum build-up that can be attained in a recharge well. Thus if other conditions are approximately equal, maximum rate of recharge may be considerably less than the rate of yield of a supply well. Under such conditions it is necessary to construct more than one recharge well to receive the discharge of a single supply well. Despite these limitations carefully constructed recharge wells, some with capacities of more than 1000 gal per min, have been successfully operated on Long Island for periods as long as 10 years without repair or rehabilitation. Even so, many operators consider it good practice to clean out or redevelop recharge wells every year or two.

Different kinds of recharge wells have been constructed on Long Island, but most are of two basic types, the so-called dry type and the so-called wet type. The former consist of perforated cased holes ending above the water table; the latter are screened below the water table or in an artesian formation. Available records indicate that about 75 pct of the recharge wells so far constructed on Long Island are screened below the water table. Selection of the most suitable type for a particular problem depends on geologic and hydrologic conditions.

Each type of recharge well has certain inherent advantages. It is commonly believed that the wet type is more successful. In the dry type of well, warm water may be discharged at a considerable distance above the water table, permitting some dissipation of heat as the water passes downward to the zone of saturation. This kind of well is usually finished with a section of large diameter pipe into which slots have been cut. The wet type of well is generally eqiupped with a standard well screen. The dry type of well, which is also shallower, is therefore less expensive to construct. However, excessive clogging is reported to take place in some of these wells. Some investigators believe the clogging is caused by release of dissolved gases as water leaves the perforated pipe and by contact of water with oxygen in pore spaces of the unsaturated sediments. There are no adequate means of reconditioning these wells, whereas the same methods used to re-develop supply wells, i.e., surging, bailing, and chemical treatment, can be applied to the wet type of recharge well. On many occasions supply wells have been used as recharge wells, continuously, or alternately as supply well and recharge well.

Depending on the type of cooling plant, the temperature of water returned to the ground on Long Island ranges from 2° to 20° higher than the temperature of water pumped from the ground. It has been shown that the return of warm water to the ground causes a regional rise in ground water temperature.[7] At some installations this rise has lowered the advantage of using ground water for cooling purposes by causing a decrease in plant efficiency. However, appreciable areal rise of ground water has occurred only in the vicinity of centers of considerable warm water recharge. Where wells are not closely spaced or are vertically separated by impervious layers there has been no serious rise in ground water temperature. Moreover, it has been observed that ground-water temperature, where raised by recharge, will generally decrease rapidly if the return of warm water is discontinued. Where an aquifer contains comparatively thin impervious layers it has been observed that there is little or no rise in the temperature of the water pumped from a supply well.[8] At a large ice plant near tide water in Brooklyn, a supply well and a recharge well, about 100 ft apart, were screened in the upper part of a gravelly aquifer containing two thin layers of clay. Shortly after recharging operations began, the temperature of the water pumped from the supply well began to rise and soon reached a temperature 20° higher than at the start of operations. The supply well was then deepened and screened in the lowermost gravelly layer. Operations were resumed and have been continued for 10 years without rise in temperature of water pumped.

But pumping from the lower gravelly layer at this plant upset another natural hydrologic balance. The supply well, when screened in the uppermost layer, produced water of low salinity. Shortly after the well was deepened, pumping began to draw sea water into the area. Eventually the chloride content in the pumping well leveled off to about 5500 parts per million, or about one fourth that of sea water.

Apparently water of this character was acceptable for compressor cooling at the ice plant, although it would not be satisfactory for many other purposes, but if the shallow well had been used as the supply well and the deeper well as the recharge well, fresh water of natural temperature probably could have been pumped continuously.

Operational experience at this ice plant illustrates how recharge wells may be used to control the temperature and chemical character of ground water if advantage is taken of favorable geologic conditions. This experience also suggests other ways in which recharge wells may be used to control the temperature and quality of ground water. One of the more obvious uses would be to lessen the danger of sea water invasion by introducing water into an aquifer at places between pumping wells and tide water, thus developing a fresh water divide between the pumping wells and the sea. Successful experimental results of this kind have been obtained by Laverty[9] along the Pacific coast west of Los Angeles.

Another allied use of recharge wells successfully practiced in Brooklyn and at El Paso, Texas, is the discharge of fresh water into a salted aquifer, from which it is later pumped at times of peak demand. Thus underground storage is utilized in place of surface storage. Artesian aquifers have been used for day to day storage on Long Island and for season to season storage at El Paso. In Brooklyn during off-peak periods, well water from public supply mains has been discharged into production wells, screened in salted aquifers. A few hours later during periods of peak demand the fresh water was pumped from storage. It is reported that about 20 pct of the fresh water injected into the aquifer was lost because of mixing with salty ground water.

At El Paso, Texas,[10] prior to 1948, heavy demands for public supply resulted in pumping rates in excess of the recharge to the aquifer, and the chloride content of the ground water was steadily increasing. To offset this condition, treated Rio Grande River water has been pumped into the aquifer for a 90-day period during four of the last five winters.

Such water management schemes cannot be effected successfully by hit or miss drilling of wells, but, as shown by elaborate experimentation at El Paso, would have to be founded on sound design and construction and careful operation.

Hydrologic trends on Long Island in the last two decades clearly indicate the danger of pumping water from wells without observing its effect on the natural balance previously existing. Prolonged delay in effecting legislation to curb excessive mining of ground water in Brooklyn, due largely to complete ignorance of the rapid lowering of water levels, has caused abandonment of all public supply wells in Brooklyn, a multi-million dollar loss. Although state control has checked additional ground water development in Brooklyn since 1933, pumpage from old wells has continued. As a result, water levels continued to decline from 1933 to 1942. Thereafter, water levels recovered slowly, but since the water table was still below sea level in much of Brooklyn, sea water continued to move into the area. Finally in 1947 the only remaining public-supply system serving ground water, a plant of 30 million gallons daily, was abandoned because of the invasion of sea water. Since then, and unfortunately only after irreparable damage had occurred, water levels have recovered rapidly.

Sometimes the aftermath of tragedy is accompanied by unexpected beneficial developments. Mismanagement of ground-water resources of Brooklyn has highlighted the possibility of using the underground reservoir in Brooklyn as an additional dry weather source of water for the New York City municipal supply system. Wet weather run-off, which now spills over the city's upstate dams, might be discharged readily into the regional cone of depression now existing in Brooklyn. Once refilled, there would be sufficient water in storage which could be pumped from the same wells to satisfy the need of the entire city for approximately 100 days, although one important factor may make the realization of this scheme difficult. After the water table was drawn below sea level earlier in the century a number of subway tunnels and other substructures were constructed in Brooklyn. It may not be possible, therefore, entirely to refill the underground reservoir without serious flooding problems.

In view of expanding industrial requirements, the search for additional sources of usable ground water will undoubtedly be intensified in the next decade. Because of this, and because much of the usable ground-water supply in the vicinity of many heavily industrialized areas in the country has already been appropriated, there is good reason to believe that many industries will be faced with costly water shortages unless wasteful utilization is replaced by intelligent water management. Much of the water now pumped from wells by industry is not consumed but is used as a heat exchange medium or a cleansing agent. Large quantities of water now discharged into rivers or the ocean could be profitably re-used if it could be economically cooled or held in storage for peak demand or dry weather use. Successful experience during the last decade has demonstrated that modification of the temperature and quality of waste water can be economically attained by recycling plant effluent through ground-water reservoirs. This, of course, does not imply that all industrial ground-water problems may be solved by the use of recharge wells. However, it is strongly evident that this method of conserving water supplies has many fruitful applications and that artificial replenishment of ground water reservoirs by means of recharge wells should be given careful consideration whenever industrial ground-water problems arise.

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aime NEWS

Ind. Min. Div. Shapes General Meeting Program

John G. Broughton, Chairman, Papers and Program Committee of the Industrial Minerals Div. announced that his group has made strong and steady progress toward organizing its share of the 1954 General Meeting to be held at the Hotel Statler, February 15 to 18.

Although papers for the Industrial Waters session have yet to be announced, preparation in this field is well along under the guidance of Pete Foose. C. F. Clausen, Chairman of the Cement, Lime, and Gypsum program has submitted a list of six papers among which are: jet piercing for drilling blasting holes; the use of cyclones for slurry-classification; report on a new type of suspension preheater now in operation at one of the Pennsylvania Portland cement companies; and a report on the latest practices in the European lime and cement industry. However, papers will not be definite until official invitations are issued. Some of the papers will be chosen for a joint session with the Committee on Raw Materials.

Two papers definitely to be presented at the Ceramic Raw Material session under the chairmanship of Lauren Wright, deal with the kyanite resources of the U. S. and with block steatite. There may be an additional paper on materials, specifications, and recent developments in the high alumina refractory field.

Six commitments have been received by M. Leonardi and R. M. Grogan, Chairmen of the Committee on Raw Materials, for a full session dealing with fluorspar. Charles Deiss reports that the Dimension Stone Committee is composed of members of top administration of marble, granite, and limestone producers. They plan a "closed" forum dealing with the broader problems of the industry.

The Division is planning a Symposium on *Careers in Industrial Minerals*. It will be a full session calling for each of the committees to contribute one paper.

Other Committees working on sessions such as Fillers, Fibers, and Pigments; Mineral Aggregates; Special Sands and Abrasives; and Rare Minerals report progress.

Outlook for the over-all technical program for the General Meeting indicates that the 1954 gathering will feature one of the top technical programs ever presented.

Notables to Speak At Fuels Conference

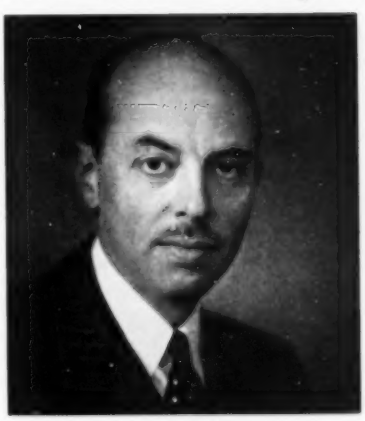
One of the outstanding features on the program of the Joint Fuels Conference, October 29 and 30 at the Conrad Hilton, Chicago, will be the luncheon meeting topics.

The first day's luncheon will see Tom Pickett, executive vice president of the National Coal Assn., discussing *Coal and World Affairs*, while on the following day Robert E. Wilson will speak on *Research in Fuels for Internal-Combustion Engines*.

The Conference is sponsored by the Coal Div. of the AIME and the Fuels Div. of the ASME. Chicago Section of the AIME and ASME are cooperating in the meeting. Registration fee has been fixed at $3.00. Luncheon tickets are $3.50 each and banquet admission is set at $6.50 per person. The September issue of MINING ENGINEERING carried the complete program.

H. W. Johnson, vice president of Inland Steel Co., will address the banquet on the *History of the Steel Industry in the Chicago Area*. F. S. Blackall, Jr., president ASME, and AIME President Andrew Fletcher are scheduled to attend and will welcome the diners. Henry F. Hebley will receive the Percy Nicholls Award for 1953 at the banquet.

Felix Wormser to Speak at El Paso Meeting



FELIX WORMSER

Felix Wormser, Assistant Secretary of the Interior for Mineral Resources, will be a speaker at the International Mining Days celebration, October 28 to November 2, at El Paso. For the first time in the history of the event, the Regional Fall Meeting of the AIME will take place in conjunction with the celebration.

Mr. Wormser will speak at the welcoming luncheon. He will be introduced by Ken Regan, Representative from the 16th Texas Congressional District.

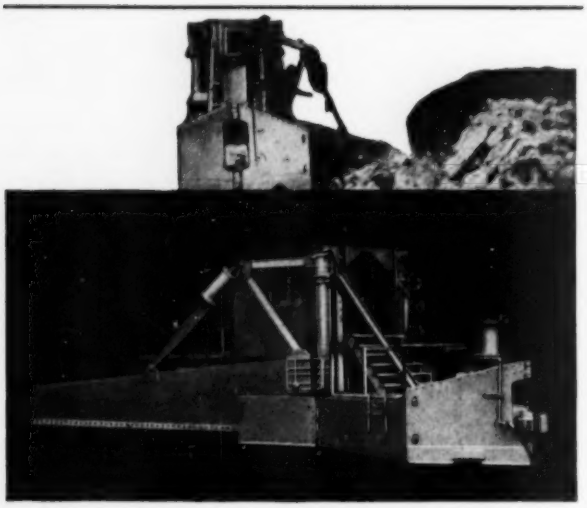
Andrew Fletcher, President of the AIME, will address a general luncheon gathering. Another speaker will be Cameron Ralston, internationally famous lecturer, who will appear at the banquet Friday, October 30.

Also taking part in the International Mining Days celebration will be the New Mexico Mining Association. Some 6000 invitations to attend the celebration have been mailed out to AIME members, while NMMA members will receive 800.

The registration fee has been set at $12.00, covering a series of luncheons, technical sessions, the Ranchero breakfast, and the suppliers' party.

Joseph P. Smith and his assistants will lead field trips to the Carlsbad Caverns and to U. S. Potash Co. and other mines in New Mexico. El Paso itself is rolling out its plushiest, reddest carpet for the expected record crowd of mining men. AIME technical sessions will center around a program including mining geology, industrial minerals, and field trips.

(Program on page 1034)



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Program
El Paso Regional Meeting

MINING GEOLOGY

THURSDAY, OCTOBER 29

Chairmen: Lloyd Nelson and Howard Quinn

1:30 to 5 pm

Fred H. Howell: *Genesis of the Climax Molybdenum Orebody*
Thomas Clendenin: *Geologic Notes on the Prieta Mine, Parral District, Chihuahua*
William Hewitt; Thomas Clendenin: *Geologic Notes on the Santa Eulalia District, Chihuahua*
William Hewitt: *Coahuila Fluorspar.*

INDUSTRIAL MINERALS

FRIDAY, OCTOBER 30

Chairmen: John Allen and Edwin McKee

8:30 am to 12 m

Leonard C. Halpenny: *Water Resources of Arizona*
Clyde S. Conover: *Water Resources of New Mexico*
George A. White: *Resume of Potash Development in the Carlsbad Area* July 1952 to July 1953
A. R. Shride: *Localization of Arizona Chrysotile Deposits*
Robert Balk: *Spurrite in the Tres Hermanas Mts., N. M.*
12 m **Joint Mining Geology and Industrial Minerals Luncheon**

SATURDAY, OCTOBER 31

Chairmen: J. P. Smith and John Wood

8:30 am to 12 m

John Allen: *Bentonitic Shales of the Upper Mesa Verde, San Juan Basin, N. M.*
George Kiersch: *Industrial Minerals Survey of the Navajo-Hopi Reservations, Ariz.*
John W. Anthony: *Gypsum Deposits of the Navajo Reservation, Ariz.*
Eugene Callahan: *Review of the Perlite Industry*
Paul Howell: *Bleaching Clays of Tertiary Age in the Southwest Quarter of the Navajo Reservation*
Wesley Peirce: *Flagstone, A Growing Industry in Northern Arizona*
Robert Wilson: *Bentonitic Clays of the Chinle Formation, Echo Cliffs Area, Ariz.*

Chairman: Eugene Callahan

1:30 to 3:30 pm

Henry A. Hulsill: *All Absorbents from Diatomaceous Earth in New Mexico*
Note: Two additional papers expected.

FIELD TRIPS

Sunday, November 1: Trip to Carlsbad Caverns under the leadership of J. P. Smith, Chairman, and assistants.

Monday, November 2: Trip through U. S. Potash and/or other mines under leadership of Joe P. Smith and assistants.

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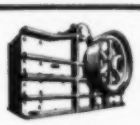
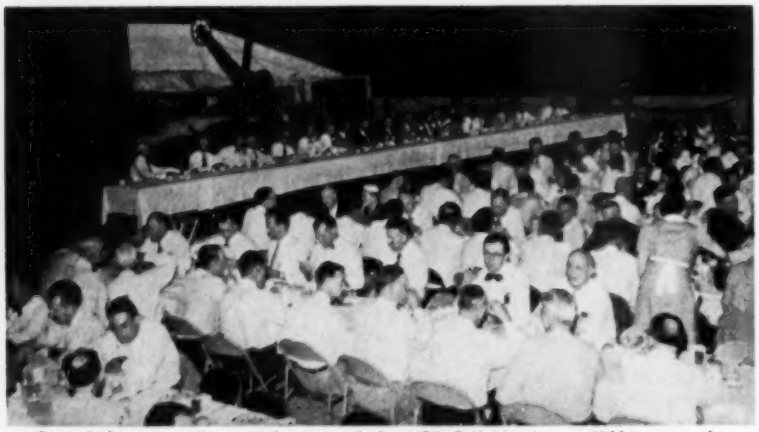

Jaw Crushers

Gyratory Crushers

Grinding Mills

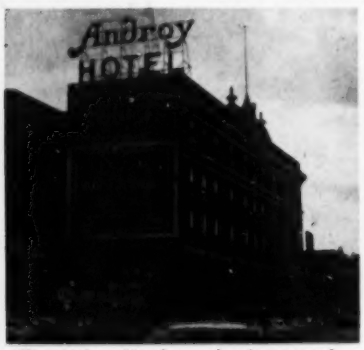
Kilns, Coolers, Dryers

Highlights in Pictures of Hibbing MBD Meeting



One of the outstanding social events of the MBD Fall Meeting at Hibbing was the luncheon held in the Hibbing Memorial Building. More than 360 persons attended, with Don Scott, MBD Chairman, acting as master of ceremonies. He introduced the heads of the Working Committees, "the men who got the work done." The Banquet and cocktail hour Tuesday evening were also held in this building.

Rudolph T. Elstad, president of the Oliver Iron Mining Div., U. S. Steel Corp., spoke on the growth of the Mesabi. Don Scott, who introduced the speaker, is on the right of the stand.



The Androy Hotel was headquarters for the meeting. Registrants were housed at the hotel and at other accommodations. The housing problem was handled smoothly by the committee.

Two registrants inspect a spiral concentrator at one of the plants on the impressive list of field trips. Field trip groups were subdivided into small sections for easier plant observation.

The State Theatre, site of Monday's technical sessions, is on the right. The banner across one of Hibbing's main thoroughfares emphasizes the city's mining industry consciousness.

EMC Advises Engineers Facing Military Service

Engineering Manpower Commission has prepared the following statement which should be of interest to young men recently, or about to be, inducted into military service, or their parents.

U.S. Army Scientific and Professional Personnel Program

The U. S. Army Scientific and Professional Personnel Program should be of special interest to recent engineering graduates who will shortly be inducted into the armed forces. It was designed to channel as many such persons as possible into Army billets which will insure the maximum utilization of their engineering or scientific training and experience. This should promote their own continued professional development and the advancement of the Army's technical programs. The program is described in detail in Dept. of the Army Special Regulation #615-25-11, dated 7 August 1952.

The regulations provide for the identification, classification, assignment, and utilization of those persons who are qualified to perform scientific and professional level duties in research and development, instruction, and related work of professional or scientific nature.

Commanders of reception centers and other installations conducting initial reception processing are responsible for the proper indentification and indoctrination of scientific and professional personnel and for their transfer to appropriate basic training.

The regulations concerning the selection of Scientific and Professional Personnel are far too complex for adequate reproduction here. The regulations differ for each major scientific and professional field. In some cases, only a B.S. degree is required. In others, a Bachelor's degree plus experience or an advanced degree is required.

Personnel qualified as Scientific and Professional Personnel are assigned a Military Occupational Specialty number (MOS) which reflects their specialty and is intended to be used as a guide in their assignment to billets in which their training and experience will be utilized. All the major engineering and scientific fields are covered by the regulation although, as has been said previously, the qualifications differ somewhat.

Copies of the basic regulation are available from the Dept. of the Army, Washington 25, D. C.

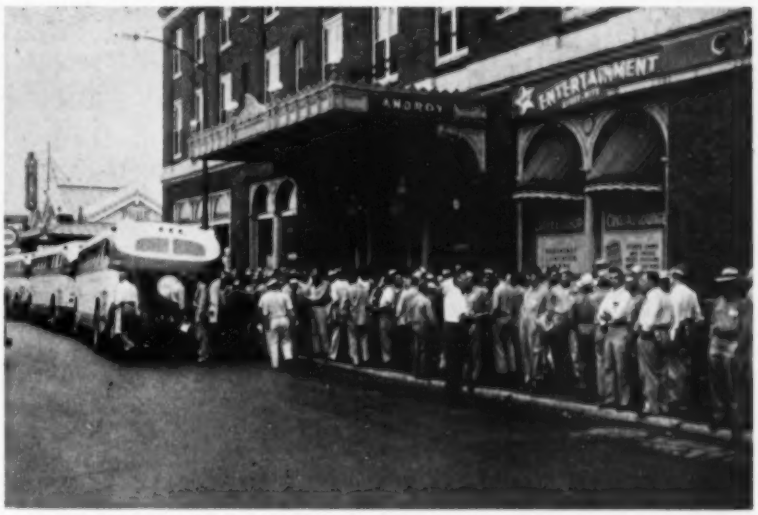
The Engineering Manpower Commission of Engineers' Joint Council is interested in the efficiency of the Scientific and Professional Personnel program. It feels that the dissemination of information regarding the existence of the program is vital to its success and would appreciate information regarding engineers and scientists qualified under the program who are not being utilized in their fields of special qualification.

Smooth Organization Typifies Hibbing Meeting

The 1953 version of the Annual Fall Meeting of the Minerals Beneficiation Division at Hibbing, Minn., proved one of the most successful on record and set a high goal for future MDB gatherings. More than 550 persons registered for the meeting which lasted from August 31 to September 2.

At preregistration, delegates were handed a kit containing maps in color of the Western, Central, and Eastern Mesabi, in addition to flow sheets and information sheets describing the plants to be visited during the field trips. The kit also contained copies of three papers to be presented during the Monday technical sessions. Reservations were smoothly handled by means of a numbering system on the preregistration envelopes.

The Monday technical session served as orientation toward understanding current methods and trends on the Mesabi range. It served to introduce the Mesabi to the men who would soon be off on field trips the next two days for a firsthand look at the plants and installations touched upon by those presenting papers. The six papers were: *Geology of the Mesabi Range As it Affects Beneficiation*, by G. M. Schwartz; *Present Trends in Mesabi Range Mining*, by B. M. Andreas; *Beneficiation of Mesabi Range Iron Ores*, by H. H. Wade; *Taconites—Their Chemical and Physical Characteristics Affecting Concentration*, by Donald W. Scott; *Iron Ore Separation in the Cyclone*, by Earl C. Herkenhoff; and *Nodulizing of Iron Ore and Concentrates*, by R. L. Bennett.



Registrants at the Fall Meeting of the Minerals Beneficiation Div. await loading time in front of Androy Hotel meeting headquarters. One of the most impressive items of the meeting was the coordination and perfection of field trip arrangements.

Donald W. Scott, MBD Chairman, served as master of ceremonies at the luncheon splitting the technical session. He introduced Rudolph T. Elstad, president of Oliver Iron Mining Div., who delivered the featured address.

The technical session was held at the State Theater where listening to the papers was easy and comfortable. Field trip organization deserves special mention. Thorough organization and tight scheduling were checked by trial runs. Each registrant received colored tickets indicating precisely where he belonged. Half the group went west, the other half east, on each day of the two-day schedule which included representative wash plant and heavy-media installations, Oliver's Extaca (agglomeration) plant, and either the Babbitt (taconite) concentrator of Reserve Mining Co., or the new Pilotac plant of Oliver Iron Mining Div. Buses carried engineer-guides and when visitors reached the plants they broke into groups of five or six under a color coding system, with a guide from the plant for each small group. More than 350 persons took in the well-rounded view of the Range.

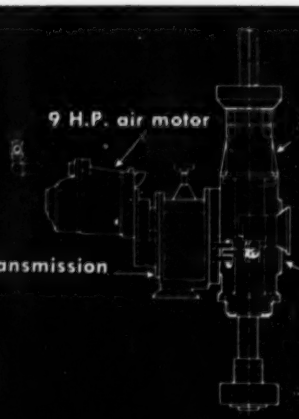
The ladies were well cared for with luncheons, sight-seeing tours, and shopping excursions. Several ladies also elected to take the field trips.

An informal cocktail hour preceded the banquet at the Hibbing Memorial Building. Warren S. Moore, president of W. S. Moore Co., served as master of ceremonies during the banquet, and L. A. Rossman, publisher of the *Grand Rapids Herald-Review* spoke to the 450 diners on "An Inspiring Chapter in the Story of Iron."

The Minerals Beneficiation Sub-Section of the Minnesota Section deserves great credit for the smoothly handled and highly informative meeting. A few of the many who shared in the job of carrying out the program under the General Chairmanship of Grover J. Holt were the heads of the working committees: W. R. Van Slyke, Technical; L. J. Severson, Finance; E. C. Herkenhoff, Housing; M. F. Williams, Trips and Luncheon; D. N. Skillings, Publicity; W. L. Taylor, Entertainment; J. R. Stuart, Registration; and Mrs. Fred DeVaney, Ladies Committee.

Field trip bus awaits MBD groups visiting Oliver Iron Mining Div.'s new Pilotac plant. Building in background houses grinding and magnetic concentration, conveyor gallery brings taconite from primary and secondary crushing sections out of picture at left.

Longyear



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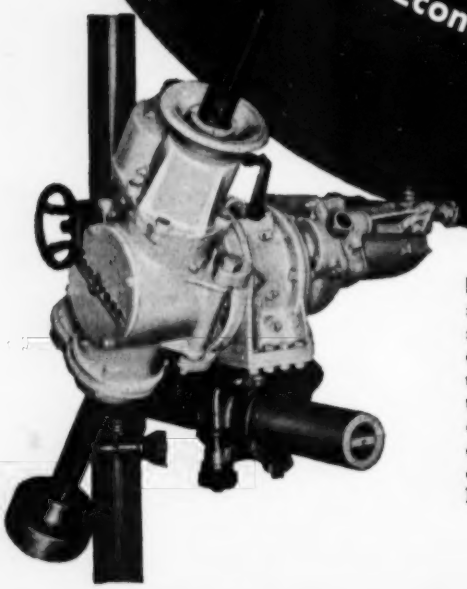

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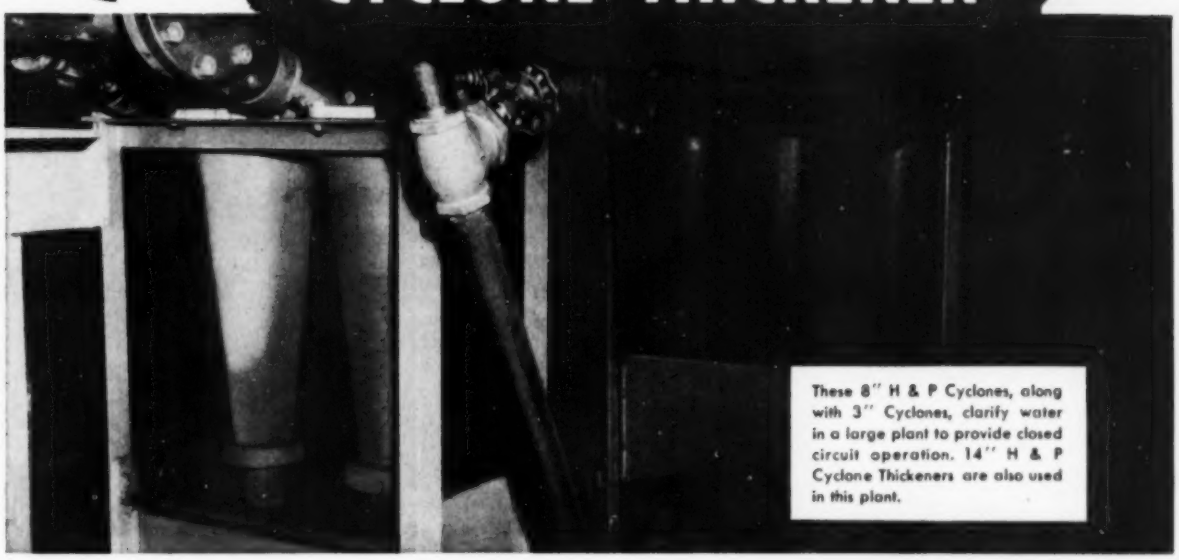

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Around the Sections

• The **Southeast Section,** with one meeting already held at Asheville, N. C., in September, is going ahead full steam with its current program, holding another gathering at Knoxville, Tenn., October 30 and 31.

• Andrew Fletcher, AIME President, addressed the **Oregon Section** at a meeting held September 18 at Portland's Mallory Hotel. A social hour and dinner were included on the program. Thomas C. Matthews, Section Chairman, has named the committees for the Pacific Northwest Minerals Conference scheduled for Portland in 1954. The **Oregon Section** has pledged its support for the nomination of Dr. Kroll for the James Douglas Gold Medal.

• S. A. Frellsen, director, Div. of Water Resources, State of Minnesota, addressed the Mining Subsection of the **Minnesota Section** at Virginia on the state's program for water conservation and what the state needs in the way of cooperation from mine operators. The Layne-Minnesota Co. presented a sound color movie, *Deep Water*, as part of the meeting program.

AN APPEAL TO THOSE WHO KNOW

We have received an appeal from a professor in one of our outstanding universities on behalf of one of his students who had completed his geology training and then was blinded in Korea. It is felt that the adjustment to this tragedy might be made somewhat easier if others in the mineral industry field who have suffered the loss of their eyesight through mine accidents or otherwise would correspond with the Korean veteran. Their accounts of how they trained themselves to new fields of activity and solved the considerable emotional problem that undoubtedly results from partial or complete blindness would go a long way toward helping this veteran who, incidentally, is the son of an AIME member. We shall be grateful for a few words from **those who know.**

Any letters sent to the Secretary of AIME at New York headquarters will be forwarded and we know will be received with grateful appreciation.

• T. B. Counselman, AIME Vice president and member of the Institute's Executive Committee, spoke on *Fluosolids Roasting at the New Carlton Mill* at a meeting of **Colorado Section.** Mr. Counselman is also Chairman of the National Membership Committee.

• Pérez Duarte, Sub-Secretary of Economy of Mexico told members of the **Mexico Section** at a meeting held in Mexico City that President Ruiz Cortinez desires to revive the country's decadent mining industry. He said that the change in the tax laws aimed at helping small and medium sized mine operators is only a beginning. Other changes in the tax structure and in the mining law are contemplated.

• **Upper Peninsula Section** held its Fall Meeting at the White Pine Copper mine, Ontonagon County, Mich. Included in the day's topics were the White Pine geology, mine, mill, smelter, general works, water supply, and townsite. Discussions took place in the morning, with the afternoon given over to touring the White Pine property.

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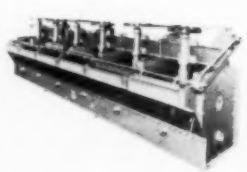

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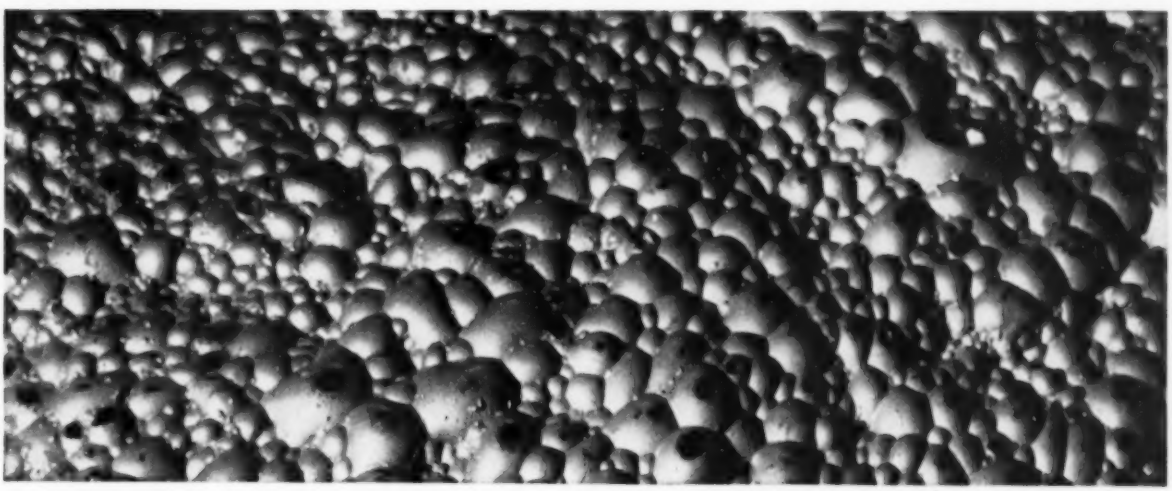

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Personals

HAROLD M. PROPP

Harold M. Propp, formerly central district manager for Nordberg Manufacturing Co., has been transferred from Milwaukee to St. Louis where he will head Nordberg's new south central district. A mining engineering graduate of the Universtiy of Wisconsin, Mr. Propp joined Nordberg Manufacturing Co. in 1941.

Clyde E. Weed, vice president in charge of operations of Anaconda Copper Mining Co. and subsidiaries, has been elected president of the Green Cananea Copper Co., an Anaconda subsidiary. **Edward S. McGlone** was elected vice president of the company. Mr. McGlone is executive vice president of Anaconda Copper Mining Co.

George R. Powe has been appointed chief mining geologist of the Northern Pacific Railway with headquarters in St. Paul.

Paul Klempner, formerly with British Guiana Consolidated Gold Fields Ltd. in Tumatumari, is assistant planning engineer with the National Coal Board in England.

Keith Whiting has been appointed to head the recently organized Toronto office of the American Smelting & Refining Co. He was in charge of exploration and engineering for the western division of A.S.&R. Co. in Salt Lake City.

Robert M. Parke has been named manager of the materials and processes section of the metallurgy research dept. at the General Electric Research Laboratory, Schenectady, N. Y. Mr. Parke will be in charge of a group of scientists engaged in pilot plant operations.

A. E. Haeseler is superintendent of mining operations for the American Smelting & Refining Co., Leadville, Colo. He was superintendent of the Mike Horse M. & M. Co. in Montana. This wholly owned subsidiary of A.S.&R. Co. ceased operations several months ago.

John W. Hough has been appointed design engineer for H. J. Daniels & Co. Inc. of Hazelton, Pa. He will lead the design work for the DMS Dense Media System in the company's project near Indiana, Pa.

Henry R. Colen, formerly with the New Jersey Zinc Co. at Gilman, Colo., has accepted a position as mining engineer with International Minerals and Chemical Corp., Columbia, Tenn.

Merlyn E. Condit has been appointed mining sales engineer in the Fairmount, W. Va., area for Mine Safety Appliances Co., Pittsburgh. He succeeds **Charles H. Myers** who has been appointed assistant to the manager of the mining department.

GROVER J. HOLT

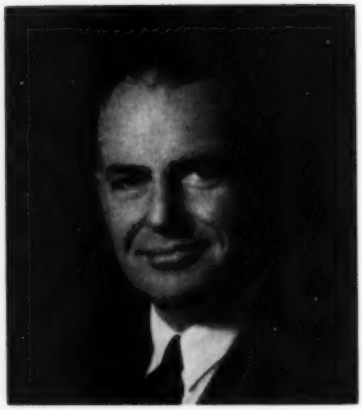
J. S. Wilbur is now vice president in charge of sales at The Cleveland-Cliffs Iron Co. **C. W. Allen** is vice president in charge of mining, land, and lumbering operations. **G. J. Holt,** formerly manager Minnesota mines at Hibbing, Minn., is general manager of the mining department, Ishpeming, Mich. His former position is occupied by **H. J. Leach.**

August V. Castelli is employed as an engineer by the Baroid Sales Div. of National Lead Co., Potosi, Mo.

Everett L. Joppa is manager of mines for Pickands, Mather & Co. in Duluth. Mr. Joppa received his M.E. degree from the University of Wisconsin in 1924 and has been associated with Pickands, Mather & Co. since 1928.

Boris V. Lubienski is mill superintendent at the Empresa Minera San Jose, Oruro, Bolivia. He was formerly metallurgist at the Colquiri Mine, Cia. Minera de Oruro.

D. K. Robinson, recently with the Camborne School of Mines in England, is with Sungei Besi Mines Ltd., Selangor, Malaya.



J. DELANO HITCH, JR.

John V. N. Dorr, founder and chairman of the board of The Dorr Company, Engineers, has announced that **J. Delano Hitch, Jr.,** has been elected president. Mr. Hitch, who was executive vice president, succeeds **E. R. Ramsey.** Mr. Ramsey is now vice chairman of the board and will also serve The Dorr Co. as a consultant.

Edward F. McCrossin, founder and president of McCrossin & Co., consulting engineering concern of New York City, has been appointed chief of the New York Ordnance District.

W. E. Remmers, president of U. S. Vanadium Co., a div. of Union Carbide and Carbon Corp., has announced three appointments to executive posts. **O. F. Holmgren** has been named executive vice president, **Arthur P. Cortelyou,** vice president, and **H. L. McKinley,** general manager.

Herbert D. Fine, mine superintendent, Frontino Gold Mines Ltd., Barrangquilla, Colombia, stopped in at AIME Headquarters in New York while on his vacation during September.

Oscar Dávila Michel has left the Castrovirreyna Metal Mines Co. in Lima, Peru, to return to his country to work for the Cia. Minera de Catavi, Catavi, Bolivia. He will do special metallurgical work.

Roy B. Earling, formerly vice president and general manager of operations of U.S. Smelting Refining and Mining Co. in Alaska, has opened an office as mining consultant at 204 Central Bldg., Seattle.

William E. Mahin, formerly director of research for the Armour Research Foundation, has been appointed technical director of Vanadium Corp. of America. He will direct the technical and research activities of Vanadium Corp. which is erecting a $1 million research center at Cambridge, Ohio, where it operates a new aluminum and ferro alloy producing plant.

James G. Hardy is general manager, Holmestake Mining Co., Winterhaven, Calif.

E. W. Bullard, president of the E. D. Bullard Co., San Francisco, has been appointed to the Research Committee of President Eisenhower's Committee on Occupational Safety, a commission established to maintain a workable standard safety code in all U. S. Industry.

M. Duhan Garretty, consulting mining geologist, of Melbourne, spent two months in midsummer visiting mining districts in Canada and the northern part of the United States, concluding with a visit to AIME headquarters before returning to Australia on Aug. 7.

R. H. RODOLF

R. H. Rodolf, formerly manager, rock drill sales, for Le Roi's Cleveland Rock Drill Div., has been named general sales manager of Le Roi's construction and mining div. He has been associated with the construction and mining industry since 1918 when he went to Vancouver to work for the Canadian Rock Drill Co. There, Mr. Rodolf worked with early detachable bits and on deep hole exploratory drilling with pneumatic drills. He joined Le Roi in 1950.

Austin Goodyear, general manager of the rubber and conveyors div. of Hewitt-Robins Inc., has been elected a vice president. Mr. Goodyear joined Hewitt-Robins in 1941.

Benj. N. Webber has returned from the Middle East and is now with the U.S.G.S. mission in Mexico City.

Jason E. Everts is now with The Eimco plant in Salt Lake City where he is in charge of preparation and production of motion pictures.

Robert W. Berkhahn, who recently joined the Western Machinery Co. as a sales engineer, is now located at 373 Old North Lake Road, Ishpeming, Mich.

Sherman A. White is general manager for Industrial Minera y Metalurgica de Oaxaca, Oaxaca, Mexico.

Emanuel Herzog, who was in Jerusalem, is now engaged in engineering work in Ankara. Mr. Herzog expects to be in Turkey for a year.

Leland C. Vought is a partner in Reynolds & Vought, Tucson, Ariz., a diamond core drilling and contract mining company. Mr. Vought was a geologist with United Geophysical Co. Inc., Tucson.

T. R. Wearing, formerly manager at Pamour Porcupine Mines Ltd., Ont., is mill superintendent at Gaspe Copper Mines Ltd., Murdochville, P.Q.

George L. Dow is with Albert Kahn Associated, Detroit. He was with Blaw Knox-Chemical Plants, Pittsburgh.

Jos. D. Doherty is with the Island Creek Coal Sales Co., Huntington, W. Va. He was Acting Chief, Synthetic Liquid Fuels Bureau, U.S. Bureau of Mines, Washington, D. C.

Harold L. Walker, head of the Department of Mining and Metallurgical Engineering, has left the University of Illinois to become a partner in the M. & N. Engineering Co. at the Taunton, Mass., branch. He will serve as technical consultant for the M. & N. Engineering Co. which is reconstructing the entire mill and establishing new procedures for the manufacture of brass products for the New England Brass Co. in Taunton.

CEDRIC H. RIEMAN

Cedric H. Rieman has been named sales manager of Gardner-Denver Co.'s mining and construction div. Mr. Rieman graduated from the University of Minnesota School of Mines in 1933 and joined the Gardner-Denver Co. in 1937.

Thomas Robins, founder and chairman of the board of Hewitt-Robins Inc., was honored on his 85th birthday, Sept. 1, by a dinner at the Wee Burn Country Club, Darien, Conn. At the dinner Mr Robins predicted, "The time is not far distant when you will see conveyors carrying coal, ore and other bulk across whole states at great speed and at enormous savings in cost."

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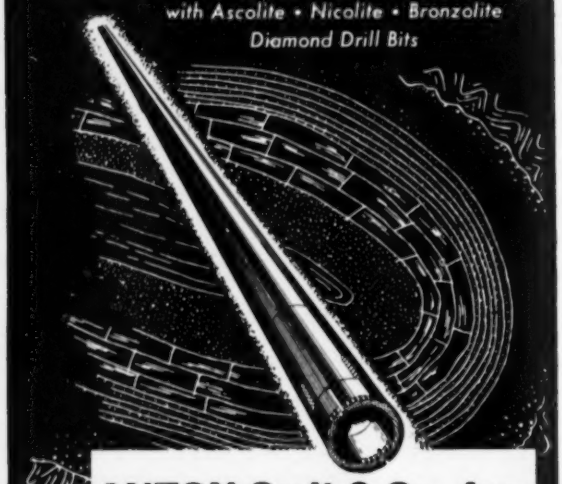

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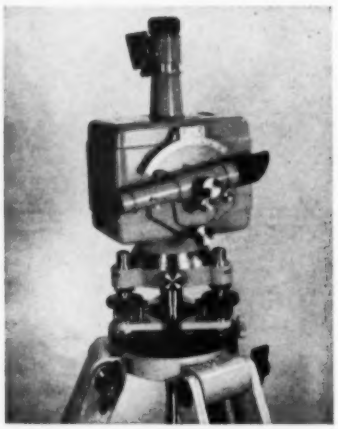

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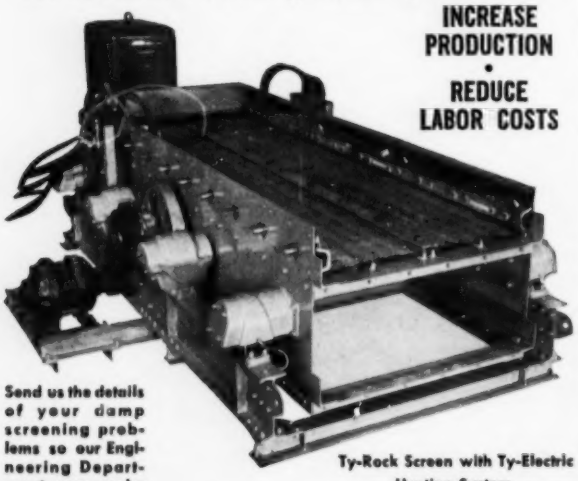

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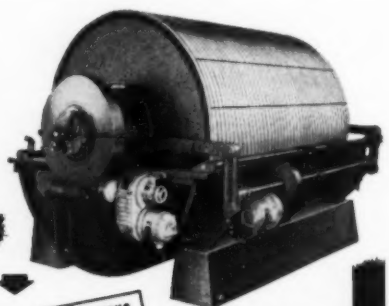

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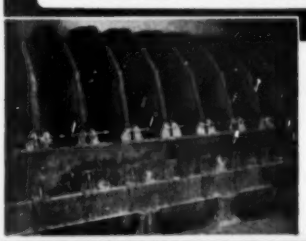

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Obituaries

Julian Boyd (Member 1905) died March 8, 1953. Mr. Boyd was a consulting mining engineer in Los Angeles. Born in Melbourne, Australia, in 1876, he studied at the Royal School of Mines, receiving the London equivalent of a university degree in mining and metallurgy. After graduation he worked as a mill manager for the Boulder No. 1. Gold Mine, Kalgoorlie, Western Australia; then transferred to Southern Rhodesia to work for the Imani G. M. & V. V. Gwanda Syndicate. In 1904 Mr. Boyd returned to Western Australia to become superintendent for Herbert Hoover, W. F. Main Reefs, Boulder Deep Levels, and Vivien Gold Mines. During World War I he saw active service as a major with the First Australian Div. From 1917 to 1919 he was Permanent-President-Courts-Martial of the A. I. F. Depots in the United Kingdom. Then, following a post graduate course at the Royal School of Mines in geology and mining, Mr. Boyd came to the U. S. to work as a mining engineer for Borax Consolidated Ltd. in Death Valley, and later for the Cactus Queen Gold Mine. After a year as an instructor of mining at the University of Southern California, he became a mining consultant. Mr. Boyd was at one time president of the Mining Association of the Southwest and president of the Los Angeles Engineering Council.

Marshall D. Draper (Member 1937), internationally known mining and metallurgical engineer, died August 14, at Beverly Hills, Calif., following an operation. Mr. Draper spent many years in China as chief engineer for the Yunnan Provincial Government at the Yunnan-Kotchiu Tin Mines, also in Burma and the Malay States. During the last 40 years he was active in examination and management of mines in the western U. S., Mexico, Central America, and Canada. Since 1930 he had resided in Beverly Hills, Calif., where he was active in mining circles of the Southwest.

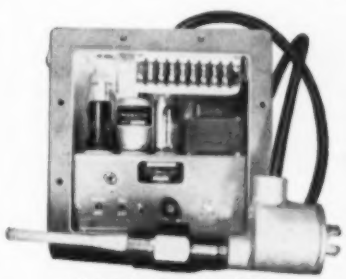
Herman Charles Forney (Member 1924) died June 18, 1953. He was a well-known San Francisco civil and mining engineer. Born in Philadelphia, Mr. Forney received his C.E. from Lehigh University in 1920. During his summers at Lehigh he worked as a machinist at the Philadelphia Navy Yard. In 1923 he graduated from Michigan College of Mines with a B.S. and an E.M. After working for various Pennsylvania mining companies, Mr. Forney in 1925 went to San Francisco where he worked as a civil engineer for the Foundation Co. of New York. He was employed for many years as a valuation engineer for the Interstate Commerce Commission and for several years as a valuation engineer for the W. R. Hearst Inventory. From 1942 to 1945, Mr. Forney served as an analyst with the War Production Board. In recent years he had been engaged in private practice with his offices in San Francisco, his home in Berkeley.

Donald F. Irvin (Member 1920) died July 1, 1953 in Bronxville, N. Y., after a brief illness. Until his retirement three years ago, he had been associated for many years with Oliver United Filters in New York City. Mr. Irvin was born in Glenwood, Mo., in 1879, and after graduating with a B.S. from the University of California in 1906, worked on the construction of a slime filter plant for Homestake Mining Co., Deadwood, S. Dak. He later worked as a draftsman in Colorado, a cyanide plant shiftboss in California, and then went to Wickenburg, Ariz., where he was acting mill superintendent for Vulture Mines Co. From 1911 to 1914 Mr. Irvin was in charge of the cyanide plant operations of the Tigre Mining Co., Sonora, Mexico, and then superintendent of a stampmill and cyanide plant for the Aguacate Mine Co. in Costa Rica. Before joining Oliver United he was engaged in the production of strontium nitrate in Los Angeles.

Kimball M. Williams (Member 1949) died Jan. 23, 1953 in Chile of polio. He was shift foreman of the Operations Div., for the Chile Exploration Co. in Chuquicamata. Mr. Williams was born in New York City in 1928 and received his early education in Pelham, N. Y. and at Mercersburg, Pa. He studied for his degree in geology at the Colorado School of Mines where he was a member of Kappa Sigma.

NECROLOGY

Date Elected	Name	Date of Death
1915	Kenneth C. Browne	July 10, 1953
1917	Kenneth H. Donaldson	Sept. 3, 1953
1937	Marshall D. Draper	Aug. 14, 1953
1950	Cedric Keith Ferguson	Aug. 11, 1953
1911	John H. Hall	Aug. 4, 1953
1952	Charles D. Kerr	June 5, 1953
1920	P. B. Lord	Aug. 9, 1953
1888	Erskine Ramsay	Aug. 15, 1953
1935	Robert Waskey	Aug. 1, 1953
1939	C. G. Williams	Aug. 14, 1953
1904	Frederick E. Wright	Aug. 25, 1953



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In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; A, Associate Member; S, Student Associate.

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Coming Events

Oct. 6, AIME, National Open Hearth Steel Committee, Buffalo Section.

Oct. 6, AIME, Washington, D. C. Local Section. Dr. J. L. Gillson, speaker. Press Club.

Oct. 7, AIME, National Open Hearth Steel Committee, Chicago Section, Phil Smidt's.

Oct. 7-9, National Assn. of Consulting Engineers, south central region, Mayo Hotel, Tulsa, Okla.

Oct. 8-9, Ninth National Conference on Industrial Hydraulics, Hotel Sheraton, Chicago.

Oct. 9, AIME, St. Louis Local Section, St. Louis University, St. Louis.

Oct. 9, AIME, National Open Hearth Steel Committee, Eastern Section, Fall Meeting, Warwick Hotel, Philadelphia.

Oct. 14-17, Twenty-first Annual Meeting Engineers' Council for Professional Development. To be held jointly with The American Society for Engineering Education. Hotel Statler, New York.

Oct. 15-17, AIME, Second Annual Clay Minerals Meeting, University of Missouri, Columbia, Mo.

Oct. 15-17, AIME National Open Hearth Steel Committee, Southwestern Section, Fall Meeting, Baker Hotel, Dallas.

Oct. 18-21, AIME, Petroleum Branch Fall Meeting, Baker and Adolphus Hotels, Dallas.

Oct. 19-21, AIME, Institute of Metals Division, Fall Meeting, Hotel Allerton, Cleveland.

Oct. 19-23, Forty-first National Safety Congress and Exposition. Sessions on industrial safety, Conrad Hilton, Congress, Morrison, and Hamilton Hotels, Chicago.

Oct. 19-23, Annual Meeting of the American Society of Civil Engineers, Statler Hotel, New York.

Oct. 22, AIME, St. Louis Local Section, Engineers Club of St. Louis and ASCE, Engineers Club, St. Louis.

Oct. 22-24, The American Ceramic Society Inc. Sixth Pacific Coast Regional Meeting, Palace Hotel, San Francisco.

Oct. 23, Assn. of Consulting Chemists & Chemical Engineers, 25th Anniversary Annual Meeting, Hotel Belmont Plaza, New York.

Oct. 28-Nov. 2, AIME, El Paso Meeting, in cooperation with International Mining Days, Hotel Paso Del Norte, El Paso.

Oct. 29-30, AIME, ASME Fuels Conference, Conrad Hilton Hotel, Chicago.

Oct. 29-31, Annual Meeting of National Council of State Board of Engineering Examiners, Plaza Hotel, San Antonio.

Oct. 30-31, AIME, National Open Hearth Steel Committee, Deshler-Wallick, Columbus.

Oct. 30-31, AIME, Southeast Section, Knoxville, Tenn. (All Sectional).

Nov. 6, AIME, National Open Hearth Steel Committee, Pittsburgh Section, William Penn Hotel, Pittsburgh.

Nov. 6-7, Annual Fall Meeting of the Central Appalachian Section, AIME. To be held jointly with the West Virginia Coal Mining Institute at The Greenbrier, White Sulphur Springs, W. Va.

Nov. 13, AIME, St. Louis Section, Coal Meeting, York Hotel, St. Louis.

Nov. 29-Dec. 4, ASME, Annual Meeting, Statler Hotel, New York.

Nov. 30-Dec. 5, Exposition of Chemical Industries, Commercial Museum and Convention Hall, Philadelphia.

Dec. 2-4, Electric Furnace Steel Conference, Netherland Plaza Hotel, Cincinnati.

Dec. 11, AIME, St. Louis Section, York Hotel, St. Louis.

Dec. 13-16, American Institute of Chemical Engineers, Annual Meeting, Hotel Jefferson, St. Louis.

Dec. 28-29, Annual Chemical Engineering Symposium, University of Michigan, Ann Arbor.

Feb. 15-18, 1954, AIME, Annual Meeting, Mining and Petroleum Branches, Hotel Statler; Metals Branch, Hotel McAlpin, New York.

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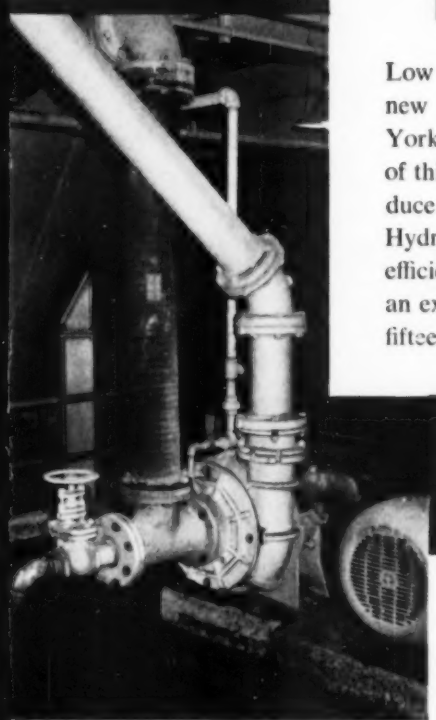
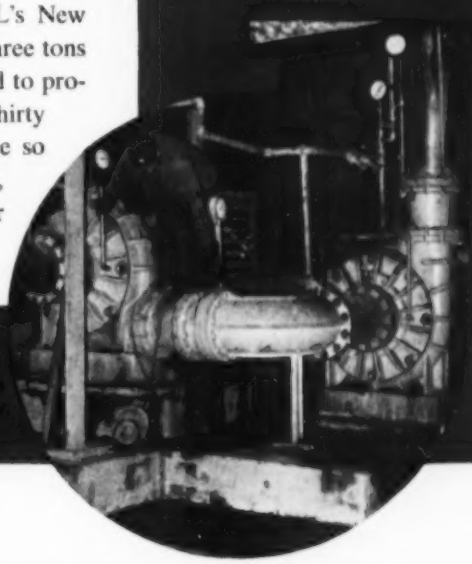
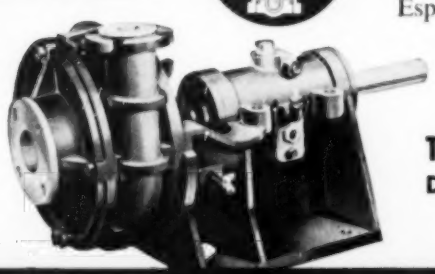






Hydroseal delivering minus-14-mesh martite ore to the spirals.

It is significant to you that Hydroseals were selected for the tough pumping at Star Lake. Especially if your operation demands economy as well as efficiency, it will pay you to query us about Hydroseals. Write today.

THE ALLEN-SHERMAN-HOFF PUMP CO.
Dept. C—259 E. Lancaster Ave., Wynnewood, Pa.
Representatives throughout the World

HYDROSEAL and CENTRISEAL
SAND, SLURRY & DREDGE PUMPS
MAXIMIX RUBBER PROTECTED

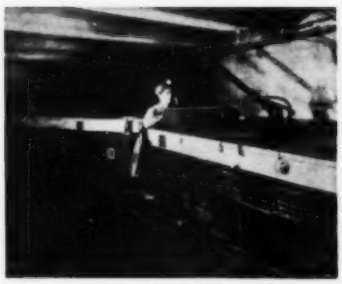
These Two M.S.A. Mine Communication Systems help you "TALK YOUR WAY" TO GREATER TONNAGE—SAFELY

THE M.S.A. MINEPHONE COMMUNICATION SYSTEM . . . answers mechanization's call for faster, safer HAULAGE OPERATIONS

Sending dispatcher's orders instantly and simultaneously to all motormen, who can reply or communicate with each other *while trips are in motion,* this modern underground communication system coordinates haulage movements with production demands . . . maintains smooth, continuous trip movements throughout the mine.

Messages, transmitted on an "open line" hook-up, keep the track clear for outgoing loaded trips and incoming empties. Keeping every trip on the move means less wear and tear on equipment; time is saved; injury is avoided.

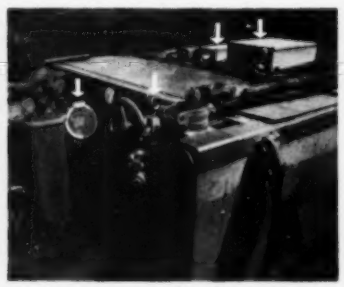
Write today for complete details on this vital production aid.



Motorman reports his position at cross-over.

Motorman, ready to go into mine, asks for right of way.

Trip unloader advises conditions at his station.

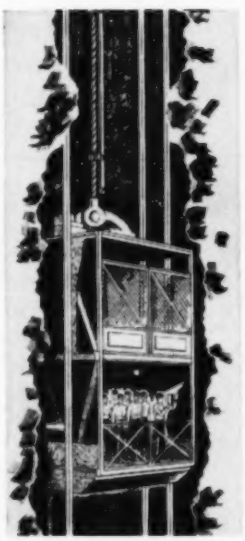


Typical compact MinePhone installation on top of locomotive.

THE M.S.A. HOISTPHONE COMMUNICATION SYSTEM . . . gives hoisting operations a lift . . . for greater production—safety

Here's the voice communication system that assures accurate, instant response between the hoisting engineer and cage. Whatever the job—load leveling—shaft repairs—shaft inspection trips—passenger transportation—the M.S.A. HoistPhone provides dependable, continuous voice communication at any level, and when cage is in motion.

The HoistPhone can be heard above noises of running equipment—is free of transmission noise and outside interference. Existing wiring, hoist cable and shaft frame can be utilized to carry the signal. It is simple to use—dependable in operation. Write for details on construction, flexibility, and design details.



HoistPhone permits positive control of the cage at any point in the shaft whether engaged in handling personnel, materials, repair or maintenance crews.

Hoist deck showing level indicator and M.S.A. HoistPhone. Voice control eliminates misunderstandings often caused by bell or horn signals.

Complete cage assembly showing transmitting and sending loop, transmitter-receiver, power supply box, microphone with control switch and speaker.

MSA
SAFETY EQUIPMENT HEADQUARTERS

When you have a safety problem, M.S.A. is at your service. Our job is to help you.

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